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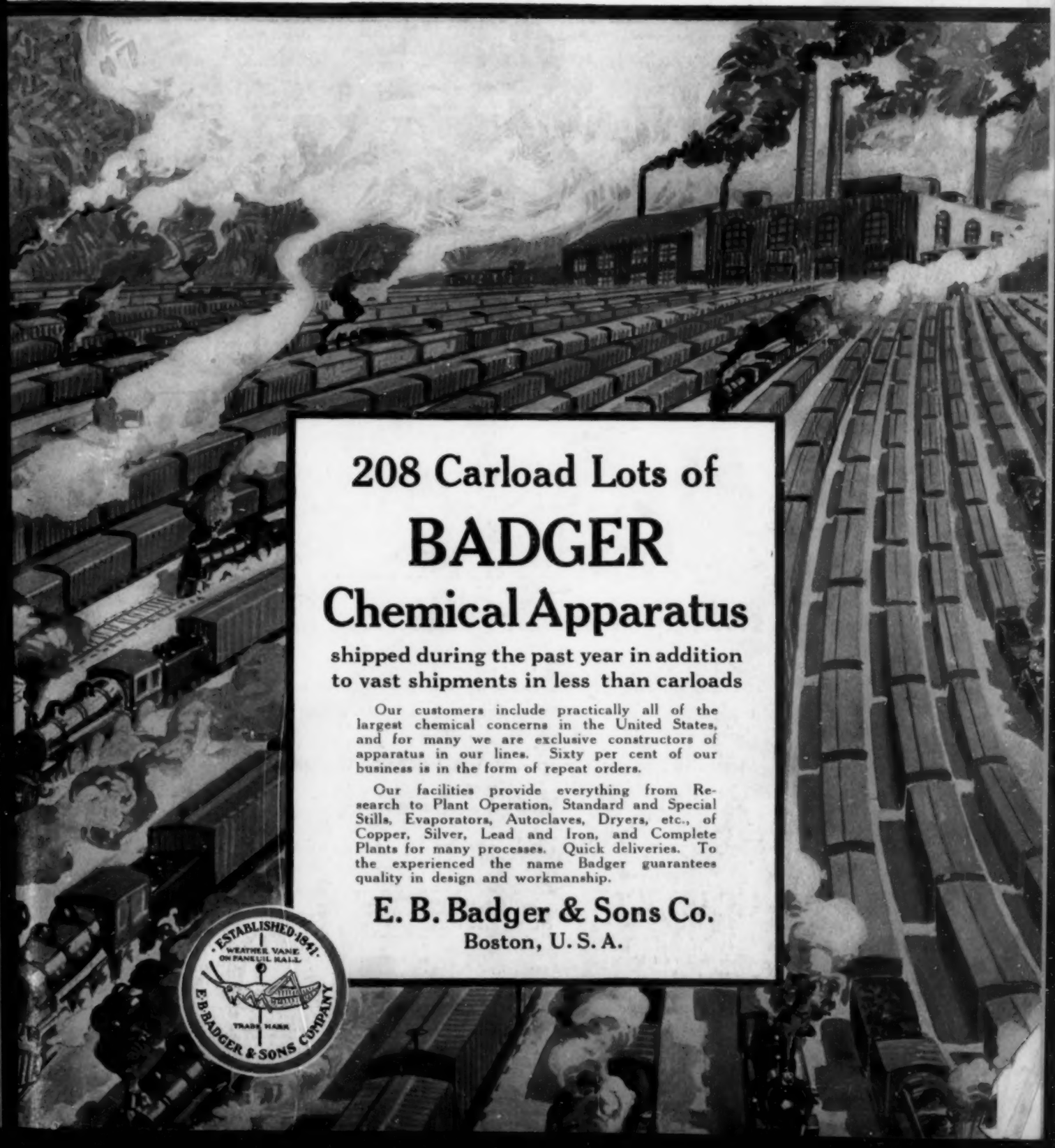

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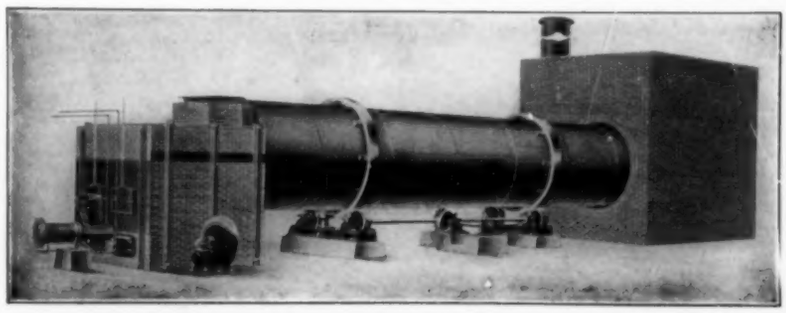

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American Chemical Industry in 1915

Heraclitus, the "dark" Greek philosopher, was born some twenty-five centuries too early for the full appreciation of the fundamental idea of his doctrine: πανται ῥεῖ. Instead of *being*, the sole actuality is *becoming* or eternal flux and change. Harmony and unity consist in diversity and multiplicity. How Heraclitus would have been charmed by the principle of relativity in modern physics. But his greatest delight would have been modern chemistry which while dogmatizing the permanence of matter, discloses matter in its frailest and most fleeting forms of existence and reveals perpetual transitory reactions where the unskilled eye sees a dull equilibrium.

The chemical industries in America to-day reflect the transient character of chemical reactions. It would be just as ridiculous to try and say what the American chemical industries *are* at this New Year's day, as to take a single instantaneous picture out of a moving-picture show and make believe that it tells the story. And all the charm would be lost which comes from the realization of the fact that we are witnesses of chemical history in the making—nay more, strong-willed enthusiastic actors and creators.

Creators, just as Luther Burbank is a creator when he grows a spineless cactus and works other marvels in plants, fruits, and flowers. For industries grow as living organisms like plants and trees. To a greater degree than plants or trees can they be shaped by the will of man, yet they always remain children of the soil. Without the natural resources, without the markets, without transportation facilities, without labor of such psychological make-up as to adapt itself to the task in hand, the most powerful will of man is powerless. Such fundamental considerations have directed the evolution of the American chemical industries, so that this country has become the largest producer of sulphuric acid, that manufacture of cellulose and wood-distillation have made great advances, and that its splendid electrochemical industries have been created of unique scope and individuality.

When the European war cut off imports and opened to American manufacturers new and clamorous markets, the whole industrial system was upset, as though a very high tariff had been enacted over night, and a fever of activity started. And here the free creative will of the American chemist came in powerfully. The American chemist saw his duty and did it there and then. He did what was humanly possible and almost more. Our readers know how the dye problem was attacked and the potash problem, how new chemical industries were started and how the old established ones readjusted themselves to changed conditions and are

straining every nerve in incessant work day and night. Look at Niagara Falls, how the electrochemical industries have been affected almost without war orders. Our Niagara friends have only one thing to worry them —the power famine. *Quousque tandem?*

While a snap shot of all this activity would be flashy enough, it would not show tendencies. We believe evolutionary sketches of different branches of chemical activity to be published during the present year will be more welcome and more useful. We begin in this issue with a sketch of the rubber industry, which, we trust, will be found very interesting, and we also publish in full Mr. Maximilian Toch's paper on the new American barium industry, though we abstracted it quite fully right after it was presented, in our issue of Oct. 15, 1915 (page 710). Mr. Toch's paper brings out clearly and strongly one all-important point. This is a glorious but a critical period in the evolution of American chemical industries. Shall they grow? Have they been built to stay? The American chemical industry is a child of the American soil. It is part of the whole industrial organism of this country. It asks the American nation for a fair attitude of mind. It needs its intelligent support, as Mr. Toch's paper states it in detail. That it may get it, let this be our New Year's greeting to the chemical industry.

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In the face of all this work with oil, comes one Louis A. Wood, of the Minerals Separation Company, with a frothing process in which no oil is used, and in which air bubbles alone do the lifting. It is yet too early to say what the effect of this invention will be, as the process is said to be a delicate one and may not succeed well enough to be applied in practice.

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Alaskan developments have reached consummation during the year and our information is to the effect that Mr. Jackling's application of copper-milling methods has been justified by results. Rolls are not to be the only machine to challenge the long maintained supremacy of the stamp mill, for the ball mill, employed years ago in Australia, is finding successful application in the same field. In the large installations of the present day the stamp mill, an inherently cumbersome machine, is at a disadvantage both as to first cost and operating cost. It is said that as compared with the ball mill it requires nearly double the mill space for equal crushing capacity and that the cost of installation is about in the same ratio. The rolls at Thane have also made splendid records, and perhaps the old machine is in greater danger of being supplanted than it ever has been before. As further invasions of its territory occur, amalgamation may simultaneously lose more ground as a primary process, since the stamp is peculiarly favored as an amalgamating device, providing at once admirable mixing of quicksilver and ore and freedom from undue "flouring" of quicksilver.

Nevertheless, such widely separated installations as the Nipissing high-grade mill and the Thane treatment of concentrates suggest that, in modified form of application, amalgamation will continue a useful tool in the hands of the well-equipped metallurgist.

Perhaps the most striking feature of all recent designing is the simplicity of the modern flow-sheet. Whether the process be crushing and concentration, as at Alaska, amalgamation and cyaniding as at the Homestake, or crushing in cyanide solution with "all sliming" as at the Hollinger and other plants, the processes and equipment are being stripped one by one of their nonessentials; and costs, responding to this analysis, are falling to levels but recently thought to be unattainable. The incomplete data available on flotation suggest that a similar winnowing is in progress here, as we note that many of the steps which Hoover's treatise deemed essentials are either being questioned or discarded. We have already alluded to Mr. Butters' efforts to avoid the use of acid; similarly the use of heat is no longer of universal application.

In cyanidation, the year has brought us little that is novel. Probably the further development of continuous counter-current decantation is the most notable feature. Patent litigation may have had its influence in hastening this development, but the process has so much to recommend it that even without the necessity of considering the question of royalties it is doubtful if filtration could in turn drive out its successor.

Several metallurgists, in discussing certain aspects of the cyanide process, have agreed that precipitation methods offer more promise of reward for the investigator than do any other phases of the process. It may be inferred, therefore, that careful study is being given to this branch of the art; yet despite this consideration and despite the abnormal prices which have prevailed in the spelter market during the past year, no developments are recorded. Zinc is still the foremost precipitant, with the dust more generally used than the shavings. Aluminium powder has a limited use. The electrolytic, sodium-amalgam and charcoal methods are not in evidence, although the first two of these are, in theory, most inviting. In a characteristically thorough paper, Professor Clevenger has reviewed and discussed electrolytic methods of precipitation.

In the laboratory we note the advent of a dependable mechanical sifting device which eliminates the personal equation and the fatigue from screen analysis. A 300-mesh sieve has also been placed on the market. Both are useful additions to equipment, for we know of nothing so informative as well made gradings of the ore in various stages of treatment, and means of facilitating such inquiries have long been needed. The 300-mesh sieve has enabled us to amend somewhat our practical definition of "slime" by withdrawing a portion of the former finest product, the minus 200-mesh material, from this classification. Slime remains the unmeasurable, the "over yonder," material.

That Vesuvius of metallurgical journalism, the discussion of the law of crushing, has been quiescent during the greater part of the year except for occasional rumblings and minor flows of equations. Quite recently, however, a tremendous eruption from the foremost Kick partisan has reminded us that now, even as in the days of the Society upon the Stanislaus, the consideration of such subjects is fraught with danger. Nevertheless, we are confident that we shall hear more of it during the coming year.

Copper Metallurgy in 1915

The past year's progress in the metallurgy of copper has been along the lines discussed in our review of a year ago. At the beginning of the European war so much uncertainty existed as to the market for copper, that production by leading companies was greatly curtailed and the price of copper dropped to a low figure. This condition continued only a short time, however, for there soon came a heavy foreign demand which resulted in a recovery in price and unusual activity among copper producers. This condition is likely to continue at least until the end of the war, and possibly for some time thereafter.

The outstanding feature of the year was the general adoption of flotation, which had the effect of discouraging experimental work in hydrometallurgy, particularly where preliminary roasting of sulphides was involved. In some cases, plans already under way for copper leaching were wholly abandoned or modified in a marked degree. The result is that developments in the hydrometallurgy of copper have not shown the strides that were so confidently predicted twelve months ago. It is true that leaching is still attractive for the treatment of certain oxidized ores, and we will hear more of this work when the extreme popularity of flotation wanes. The process of Weidlein, previously mentioned in our columns, is under experimental operation, and apparently offers promise of successful development.

Flotation has found its greatest economic use in the recovery of finely divided mineral. Where the copper

minerals occur in small particles finely disseminated through the gangue, necessitating fine grinding for their liberation, or where friable copper minerals occur with other less friable sulphides, flotation is admirably adapted as a process of concentration. If, in addition, oxidized minerals accompany the sulphides, the former also may be floated after treatment with a soluble alkaline sulphide, coating or filming the particles with copper sulphide. This effects a greatly enhanced saving. Past losses in concentrating have amounted to as much as 30 and 40 per cent, due mainly to an excessive loss of the finely divided mineral; but this will be largely overcome by the use of flotation which yields a high percentage recovery on slime. The process also seems admirably adapted to the treatment of the "porphyry" ores as an initial step.

On the other hand, ores like those of the Butte district, which yield good products with jig and table concentration, require flotation simply as an auxiliary step after the use of the older standard methods. At the same time, having lost the fear of being unable to make a satisfactory recovery on slime, the upper limit of size for jigging has been greatly reduced, with the result of a greater total recovery of metal through the use of flotation. Tailing formerly carrying from 0.5 to 1 per cent copper may now be reduced to about 0.1 per cent.

The production of fine flotation concentrates has had the further effect of necessitating a greater extension of reverberatory smelting. Agglomerating methods applied to fine copper mineral to fit it for blast-furnace smelting have not been extensively applied, and it is fair to assume that by far the major portion of all fine concentrates will in the future be smelted in reverberatory furnaces.

The new furnaces, as previously noted, are much larger than the earlier types, and will permit a continuous flow of slag. The new method of charging along the full length of the side walls also renders unnecessary the frequent fettling called for under former practice, because the charge banked up against these walls affords a protection from slag corrosion.

The present tendency is toward coal-dust or oil firing of reverberatory furnaces, depending on the location of the plant with respect to fuel supply. In some localities coal of any grade is expensive while oil is relatively cheap, and under such conditions it is doubtful if coal-dust firing can find favor. Excellent fuel ratios are obtained under either system. Other advantages are found in an increased tonnage; maintenance of a higher temperature; requirement of less draft than with ordinary grate firing.

The requirements for oil-firing are comparatively simple. The oil is preheated to a temperature ranging from 160 deg. Fahr. to 200 deg. Fahr. at different plants, and is fed under a static pressure of from 35 to 40 lb. In the burners the oil is atomized by air at 40-oz. pressure.

Coal-dust firing is not new in its inception, but early trials were failures because many of the features essential to success were not understood. These features are: drying the coal to 1 per cent or less moisture; fine pulverization, so that 93 to 97 per cent will pass 100-mesh, and 82 per cent is finer than 200-mesh; proper proportioning of fuel and air; use of fuel with 20 to 30 per cent or more volatile matter; correct design of the furnace and provision for taking care of the ash formed.

While blast furnaces are being discarded in certain plants, they are indispensable where low-grade high-sulphur ores are available, as at Mt. Lyell, Tasmania, and where pyritic or semi-pyritic smelting can be practised they bid fair to remain in use as long as suitable ores can be found.

In the realm of crushing and grinding for concentration there has been a steady development of ball mills of various types. The principle feature of their construction is a lowering of the point of discharge as compared with that of feed. In some case this is accomplished by means of lifting devices in the discharge end, while the extreme suggestion is for the use of an open-end discharge. Disk crushers and grinders also have been receiving favorable consideration for both coarse and fine work. These machines offer the advantage of giving a true crushing action without grinding by abrasion also, and for certain purposes this effect should be valuable.

The Year in Zinc Metallurgy

The logical sequence of the unusually high price for spelter during 1915 was a stimulus, not only to ordinary retort smelting but to the development of other methods of reduction and the utilization of low-grade ores. Smelting plants which had been abandoned for years were rehabilitated, and some new construction was completed. The Donora plant of the United States Steel Corporation was brought into production in the record time of four months and ten days from the time of beginning construction, or in about one-fourth the normal time for the construction of such a plant. An unusual feature of this smelter is the extensive use of concrete in its construction.

The Joplin district profited tremendously from the high prices which prevailed, and the construction of new concentrating plants has exceeded any previous record. The later plants are of good design and should effect a higher saving, particularly on slime, than has been customary in the older mills.

In the field of dry concentration and separation, there have been no new developments beyond the standard electrostatic and electromagnetic machines and the Plumb pneumatic jig, all of which are now well known.

During a year which gave promise of definite results from electrothermic smelting, there was little or nothing forthcoming, and at the time of writing much more interest centers in electrolytic zinc. The present position of these two processes is quite reversed from that of a year or two ago. At that time electrolytic methods were not received with much favor by some of the largest zinc-producing concerns on account of the high capital cost for plant, whereas electrothermic smelting was conceded to be promising for the treatment of

mixed sulphide ores of zinc, copper and lead. To-day, although electrothermic reduction still holds its possibilities, it is not as near commercial reality as is the other process. Some of our largest and most progressive metallurgical organizations have undertaken the leaching of zinc ores, with electrolytic deposition, and have experimental plants under operation. The article by D. A. Lyon, O. C. Ralston and J. F. Cullen on hydrometallurgy of zinc and lead in 1915, published elsewhere in this issue, will be found particularly interesting in this connection.

The zinc oxide plant at Leadville, already described in this journal, has afforded an outlet for a small tonnage of low-grade carbonate ore. The product has been in steady demand, finding its way into the chemical trade.

Calcination of zinc carbonate ore for the purpose of enriching it by expulsion of carbon dioxide is not as extensively practised in this country as in Mexico. In the latter country the process has been perfected to a satisfactory point and is in operation at several places. In the United States the only calcining plant of which we have knowledge is that of the Empire Zinc Company, in Nevada. Coarse ore is calcined in a vertical kiln, and the fine in a rotary furnace. The success of the process in Mexico seems to be due to the peculiar form of kiln used and the careful manner in which the charge is fed by hand. For the latter purpose cheap labor is an essential.

Needless to say, flotation has played a conspicuous part in the concentration of zinc ores, greatly increasing the recovery. Experimental work on the flotation of carbonate ores is proceeding, with better prospect of success than could be reported a year ago. Just how much zinc has been added to our total output as a result of flotation is difficult to estimate, as the use of the process has so modified the previous milling practice that the mineral actually recovered by flotation is probably much more than that which was counted a loss in former practice, and which flotation was first expected to save. In any event, the total must represent an appreciable percentage of our total production.

Chloridizing and other special processes have been quiescent during the year, but their exponents are still active and promise developments in the near future. At the present time active experimentation is a negative quantity.

The Steel Trade's Wonderful Year

To reach a correct conception of the influence responsible for the remarkable showing made by the American steel industry in 1915, its change from a rate of operation less than 35 per cent of its capacity to a rate of operating under the highest pressure it ever knew, the most direct method seems to be first to analyze the character of the demand at the close of the year, to consider the precise position attained before endeavoring to trace the steps. One reason for adopting this course is that during the year the impression arose that steel activity was due in large part to orders for "war steel" and while that view may have been substantially correct at one time it was not correct as to conditions in the late months of the year. If the influence of orders for war steel is rated too highly, the easiest method of correcting the misapprehension is to consider conditions in the steel market at the close of the year.

Steel, of course, is involved in two classes of exports, the steel exported directly from the mills and reported by weight in the government's export statistics, and the steel which passes to various manufacturers and is converted into merchandise that is exported. The iron and steel exports returned by weight increased steadily during the year until August, when they reached a maximum of 406,000 gross tons. There was a decline to 382,000 tons in September and a further decline to 351,000 tons in October. Excluding from the October returns pig iron, scrap, iron castings, etc., there was about 300,000 tons of finished steel, representing a rate of 3,600,000 tons per annum. Steel production increased from August to October, so that the percentage of steel exported decreased very materially. Steel shipped by mills for indirect export can be estimated only very roughly. It includes steel for shells, arms, locomotives, freight cars, automobiless, machinery, etc., but an estimate of 300,000 tons a month seems high. If we assume that the total steel made in October for export, direct and indirect, was at the rate of 7,000,000 tons a year, it represented no more than 25 per cent of the production, which may safely be estimated at about 28,000,000 tons a year, in terms of finished rolled steel. The remainder passed into purely domestic use. In November a serious freight congestion began to develop between mills and tidewater, resulting early in December in the railroads placing embargoes on shipments of steel intended for export. Shipments of the mills for export were thereby reduced, while shipments to the domestic trade were correspondingly increased, and yet the domestic trade was not glutted, but called for more steel. Thus the condition at the close of the year was that steel-mill shipments represented something like 80 per cent domestic and 20 per cent export, direct and indirect. A portion of the export material, furthermore, was not war material.

The domestic steel-consuming trade had thus become a very large buyer. There were two chief causes for the remarkable improvement that occurred during the calendar year 1915, one being that in the closing months of 1914 industry had passed far below normal and could not but react, the other being that the tremendous export trade of the country was making it prosperous. The war produced a mental panic, which reached its worst stage perhaps at the close of October, 1914. There would have been more than a mental panic if the stock exchanges had not been closed and the new banking system had not been scheduled to go into effect on Nov. 16. While industry was generally depressed, the buying of steel reached a lower ebb than corresponded to the amount of industrial activity, because jobbers and manufacturing consumers reduced their stocks. In our review of a year ago, explaining the extreme depth of depression to which the steel trade had fallen, it was pointed out that fluctuations in steel production were

due in considerable part to the fact that in this industry the buyer rather than the seller carries stocks.

In December, 1914, a moderate amount of steel business was booked, at prices which averaged, for the various finished products, a somewhat lower level than that which obtained in November, 1911, and were therefore the lowest in more than 15 years. The mills were operating at less than 35 per cent of capacity. The business booked was chiefly for January and February shipment. The mills then succeeded in mustering sufficient courage to advance bars, plates and shapes from 1.05c. to 1.10c. on Jan. 1, a proceeding which gave some encouragement to the trade at large. By very easy stages other price advances were made during the half year. It was recognized that the preceeding was somewhat unusual, for the history of American steel markets in general shows that prices have no inherent strength, or at least advancing tendency, unless the mills are operating at about 85 per cent of their full capacity.

A remarkable part was played by "war orders" for steel during the second quarter of the year. While such orders had been placed at intervals the bearing upon the market as a whole had not seemed important. In April reports were circulated of very large tonnages of shrapnel rounds being sought, both for direct export and for shell manufacture in the United States. In many quarters these reports were not taken very seriously. Secrecy surrounded the placing of many of the orders, but in the course of a few weeks domestic buyers of steel came to realize that the war business really was important. In May and June orders were placed for fully 50,000 freight cars, about one-third being for export, these orders being more than double the total placed in the nine months from the inception of the war up to that time. The fear that the war demand for steel might make steel scarce in the United States, the encouragement given by the resumption of railroad buying, and some minor influences, gave a very decided stimulus to steel demand in general. Steel mills operated at close to 85 per cent of capacity in July and in August they reached substantially full operation. Deliveries began to fall behind and buyers then took hold with vigor.

Steel prices, which advanced rather haltingly during the first half of the year, and then only by an average of about $2 a ton, began to advance more rapidly, and in October an unprecedented rate of advance was developed. Steel prices rose during the last three months of the year at an unprecedented rate, barring only the altogether exceptional advance of 1899.

The mere volume of demand was not sufficient to account for the exceptionally rapid advance in these three months. There was an abandonment of the policy pursued by steel manufacturers in various other periods of heavy demand, that of holding prices down to what was regarded as a fair normal level. The decision rendered June 3, 1915, in the case of the Federal government against the United States Steel Corporation had given the corporation itself a clean bill of health, while it had roundly condemned the "Gary dinners." The steel makers felt that they were perfectly free to allow prices to advance indefinitely, if they tended to do so; that as they were denied the privilege of holding them up to what they considered a normal level, in times of poor demand, there was no occasion to hold them down to such a normal level in times of good demand. There was the further influence that the demand for steel was regarded as largely a creature of the war, that a termination of the war might greatly reduce the demand. The advances in finished steel prices during the second half of the year averaged about $11 per ton, making about $13 advance for the whole year. The advance could be regarded as still greater if account were taken of the premiums paid for prompt deliveries in the case of some products. The level at the close of the year was a trifle above the top point reached in 1907 and therefore represented the highest level since 1902. Prices were about $6 a ton above the top reached in the 1912 movement and about $4 a ton above the highest level reached in the movement of 1909.

As already indicated there was a decided increase in domestic buying in the mid months of the year, due in considerable part to sentimental influences. Such buying would in the course of months play out; buyers would become stocked with material, but no such playing out has occurred. No stocks to speak of seem to have been accumulated; on the contrary many buyers are clamoring for better deliveries. The actual consumption of steel has evidently greatly increased.

For many decades, through the year 1907, the demand for iron doubled on an average once in each ten years, though far from regularly year by year. It becomes impossible to judge at any time what is the normal demand, to determine whether a given demand being then experienced is normal, below normal, or above normal. There was a period of twelve consecutive months ending October, 1907, in which the production of pig iron was 27,000,000 tons. The existing capacity, eight years later, is 38,000,000 tons or thereabouts, representing a considerably smaller increase than would be represented by a doubling in ten years. A demand that fully engages the existing capacity may not be an abnormally heavy demand at all. It is a fact that a considerable number of the steel consuming industries are but indifferently employed. The demand for structural steel, for instance, has been conspicuously light.

Perhaps it is no great measure of prosperity in the consuming trades that has made the steel industry itself so prosperous. The prosperity of the country at large is traceable to the enormous balance in its merchandise trade. In the twelve consecutive months ended with November the merchandise trade balance in favor of the United States exceeded $1,700,000,000. For a number of years an apparent favorable balance in merchandise of about $500,000,000 served to equalize the unseen balance. A balance $100,000,000 a month greater naturally induces prosperity.

Pig-iron production in the United States was 12,233,791 gross tons in the first half of 1915 and about 17,700,000 tons in the second half, making about 29,900,000 tons for the year. The record calendar year was 1913, with 30,966,152 tons, the record half-year being the first half of 1913, with 16,488,602 tons to its credit.

Readers' Views and Comments

Nomographic Charts for Conveyor Belt Calculations

To the Editor of Metallurgical & Chemical Engineering:

SIR:—I have studied with much interest the chart for the quick calculation of problems relative to rubber conveyor belts which was published in the Nov. 1, 1915, number of METALLURGICAL & CHEMICAL ENGINEERING. The graphical method of representation is finding new applications every day and Messrs. J. D. Mooney and D. L. Darnell have given us a very good example of the rectangular co-ordinate type of chart. It has appealed to me, however, that the equation in question is admirably suited to another type of graphical solution, namely, the nomographic chart, and with the idea of calling attention to this second graphical method, I have constructed the charts, reproduced on page 9.

Fig. 1 is a nomographic chart for the solution of the same problem as that treated by the aforementioned authors, namely, the determination of the correct number of plies for a rubber conveyor belt operating under certain conditions. The equation represented is given by them as

$$p = KgW(L + 10H)$$

where

p = the correct number of plies.
K = a constant depending on the type of drive. For a simple drive with a bare pulley, it has the value 1/250,000.
g = the weight in pounds per cubic foot of the material handled.
W = the width of the belt in inches.
L = the length of the belt in feet (approximately twice the center distance).
H = the difference in elevation between the head and tail pulleys in feet.

The nomographic chart consists of three scales and one supporting line BB. On the scale AA is the length factor f equal to $(L + 10H)$, on the scale CC, the width of the belt W and also the weight of the material g, and on the scale DD is the number of plies p. To solve the equation for p, knowing the other variables: Place a straight edge across the chart connecting the points representing W on the scale CC and f on the scale AA; place a pencil point on the intersection of this line with BB and swing the straight edge till it connects this point with the point on CC representing g; where this line intersects the DD scale read off the correct value of p.

For example, take the same set of values used to illustrate the rectangular chart, width 36 in., length 300 ft., difference in elevation 20 ft. and material, sand and gravel. The arrow pointed lines show the solution of this problem. Connect a (500 on AA) with c (36 on CC); connect b, the intersection of this line with BB, with c (90 on CC) and note that d on DD falls between 6 and 7 indicating that 7 is the proper number of plies. Note that it is unnecessary to actually draw the lines connecting the points.

The advantages of the nomographic as compared to the rectangular co-ordinate chart are briefly these:

1. It is easy to construct; but few lines are necessary to represent a formula and the calculation is simple.

2. It is easy to read; it does not present a mass of lines to the eye.

3. It is easy to interpolate; interpolation is always easier on a scale than between lines on a plot.

4. The solution is equally simple whichever quantity is the unknown.

5. Any number of variables can be accommodated.

The rectangular chart for the solution of the formula was for but one drive only, and the value obtained by it had to be multiplied by a conversion factor if a calculation for one of the other types of drive was desired. The nomographic chart can be made to accommodate the added variable by merely adding another scale and line. Fig. 2 shows this chart arranged to accommodate the varying K. To solve the same problem as before, connect a (500 on AA) with c (36 on CC); connect b, the intersection of this line with BB, with c' (90 on CC); connect b', the intersection of this second line with $B'B'$, with a' ($K = 1/250{,}000$); this line cuts out the correct value of p on DD.

The nomographic chart is especially applicable to exponential and power equations and a study of its utility in this direction is well worth while.

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What Is the True Value of a Complex Ore?

To the Editor of Metallurgical & Chemical Engineering:

SIR:—From time to time I have noticed articles by many different engineers and metallurgists on the subject of "Mine Valuation." I do not recall having seen anything in the nature of a discussion of "Ore Valuation." Since my work for the past thirteen years has been almost entirely along the lines of valuation of complex ores, especially those containing zinc, I propose to open a discussion by submitting the following:

Having worked out the treatment of several "complex ores" and having attempted to devise a treatment for several hundred others, I have taken a specific case which has come to my personal attention and which, I believe, will serve the purpose of argument more completely perhaps than a general discussion of the subject.

During the greater part of the present year, owing to the unusual activity in the zinc industry, engineers have had many occasions to examine and report on mines, the values of which depend to some extent on the zinc contents of the ores. The mine is sampled and the ore assayed and measured to a point where the engineer finds a certain number of tons carrying a certain amount of gold, silver, copper, lead, zinc, iron and any other metals that can be recovered and sold. It is at this point in his examination that an understanding of the methods of treatment to be employed that will effect a commercial saving of the metals, the cost of such method and the marketing of the products, is fully as essential to the correctness of his findings as is the knowledge of methods for sampling, assaying and measuring the ore.

It is at this point also that the assayer has found himself in a most embarrassing position; not on account of his inefficiency as an assayer, but because he has not the facts at hand upon which to base a sound opinion as to the value, in dollars and cents, of the ore upon which he has determined the metal contents. I have seen assay certificates from reputable assayers which showed fully the metal contents, and in the space for that purpose was shown a value which had been calculated from the analysis. In many cases these "values" have been found to represent the metal values, based upon the current market quotations for the dif-

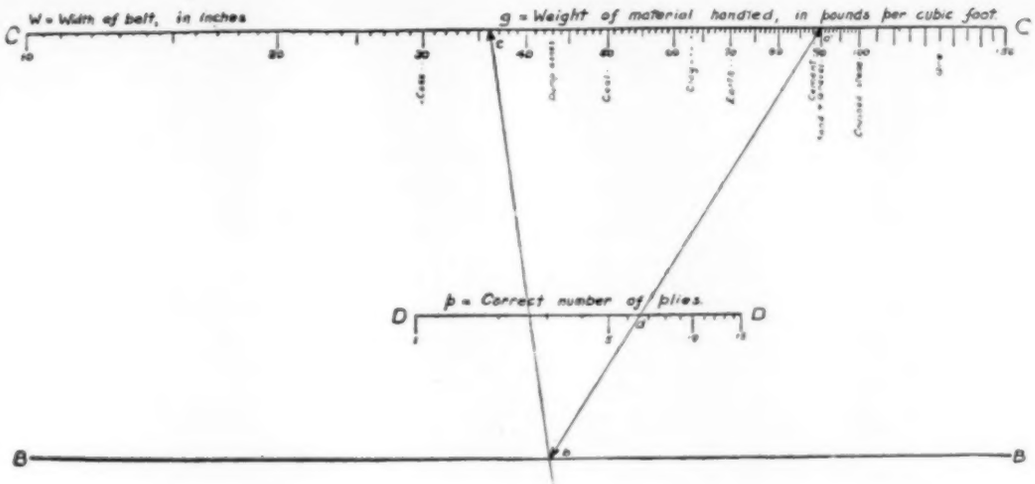

FIG. 1—CHART FOR CONVEYOR-BELT CALCULATIONS, CONSTRUCTED BY R. E. HAYLETT

$$p = kgW(L + 10H)$$

FOR A SIMPLE DRIVE WITH BARE PULLEY

$$K = 1/250{,}000$$

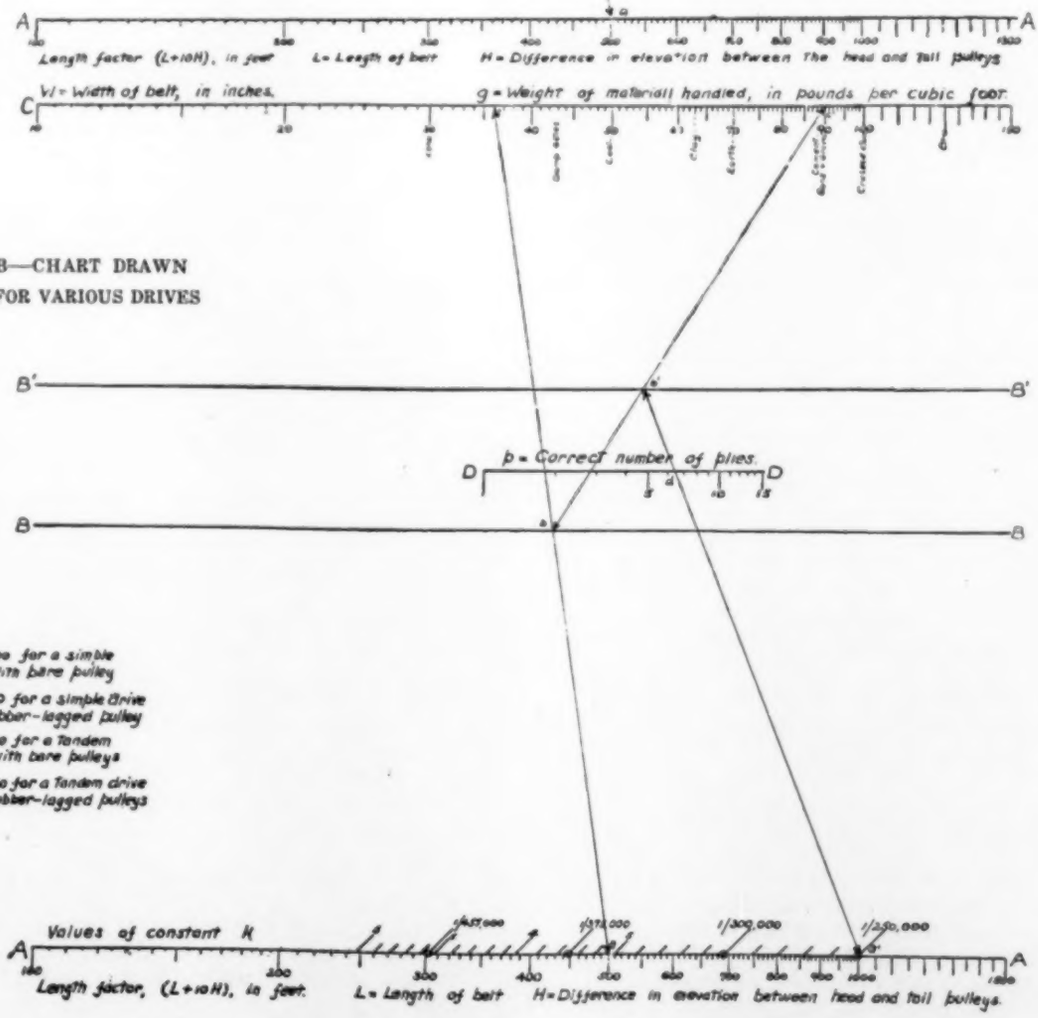
Length factor (L+10H), in feet
L = Length of belt
H = Difference in elevation between the head and tail pulleys
W = Width of belt, in inches
g = Weight of material handled, in pounds per cubic foot
p = Correct number of plies
K = 1/250,000 for a simple drive with bare pulley
K = 1/300,000 for a simple drive with rubber-lagged pulley
K = 1/375,000 for a tandem drive with bare pulleys
K = 1/450,000 for a tandem drive with rubber-lagged pulleys
Values of constant K
Length factor, (L+10H), in feet
L = Length of belt
H = Difference in elevation between head and tail pulleys

FIG. B—CHART DRAWN FOR VARIOUS DRIVES

ferent metals delivered in New York. It is needless to say that this method is misleading and may result in great financial disaster if further investigation is not undertaken.

Many engineers have attempted to estimate such ore values by allowing "liberal" losses. By this method great mistakes may be made chiefly because the losses allowed have not been sufficiently liberal. It is my intention to try to show that there is no possible way by which these losses can be estimated. They must be accurately determined, and in most cases, they will be found to be surprisingly large.

Some months ago I was asked by an engineer who had finished an examination, what percentages of saving could be reasonably expected by the most modern practice. I was told that the ore had a value of $12 per ton, based on a price of $20 per ounce for gold, 50 cents for silver, 50 cents per unit for lead, and 60 cents per unit for zinc. He had not assayed for iron since it had little value. I was unable to make an estimate for I did not know, nor did any one know, what grades of products could be made or how much of the gold and silver would be found in the zinc product. These facts, which are absolutely essential, can be determined and must be determined before the value of the ore can be definitely fixed.

To illustrate how difficult it would be to estimate a value of this kind, I submit the following table to show what products can be made and what money will actually be received at a certain stage of the metal market, for a certain ore.

	Wt.	Oz. Au	Oz. Ag	Per Cent Pb	Per Cent Zn	Per Cent Fe	Per Cent SiO_2
Original ore.....	100	0.07	9.00	8.00	16.00	7.00	46.00
Lead product	8	0.32	45.00	68.00	5.00	6.00	2.00
Per cent saved.....	..	36.50	40.00	68.00	2.50	6.80	0.30
Iron product.....	14	0.22	17.00	8.00	11.00	32.00	6.00
Per cent saved.....	..	44.60	26.40	14.00	9.50	64.00	1.80
Zinc product.....	26	0.02	6.50	2.60	48.00	5.00	10.00
Per cent saved.....	..	8.50	19.00	8.60	78.00	18.50	5.60
Slime	7	0.04	11.00	7.00	12.00	6.50	49.00
Per cent saved.....	..	4.00	8.50	6.10	5.20	6.50	7.40
Tailing	45	0.01	1.20	0.60	1.70	0.70	86.00
Per cent lost.....	..	6.40	6.00	3.30	4.70	4.20	84.90

It will be seen that the total concentrate carries 89.6 per cent of the gold, 85.5 per cent of the silver, 90.8 per cent of the lead and 90.1 per cent of the zinc. This can certainly be called a high saving.

The original ore has a value of $29.10, estimated upon metal contents and quotations, with gold at $20 per ounce, silver at 50 cents, lead at 4½ cents per pound and zinc at 5 cents. Estimated with gold and silver at the same value as above, and lead at 50 cents per unit and zinc at 60 cents per unit, it has a value of $19.50. When the products are sold, however, they will be sold about as follows:

In lead concentrate we will receive $19 per ounce for gold, 47½ cents for silver, 3¼ cents per pound for 90 per cent of the lead (wet assay less 1½ per cent), 10 cents per unit for excess iron. We will pay $3 freight and no treatment charge. This leaves us $63.75 per ton net, or $5.10 per original ton.

Iron concentrate will bring the same prices for gold, silver, lead and iron. We will pay same freight and about $5.50 for treatment. This leaves $11.31 per ton net, or $1.57 per original ton.

The zinc concentrate, with spelter at 5 cents, and $5.50 freight, will net $19.30 per ton or $5.02 per original ton.

These net receipts amount to $11.69 per original ton, which is 31 per cent of the assayer's figure and about 60 per cent of the engineer's.

A careful analysis of all of the above figures will show that we have received net, $11.38 per ounce for gold, 25.2 cents per ounce for silver, 2.09 cents per pound for lead, and 1.57 cent per pound for zinc, which figures may be safely applied to this particular ore so long as the grade does not change materially. The contracts upon which I have figured settlements may vary to some extent, but the general result will be about the same.

From the above it will be apparent to the investor and to his engineer that the metal contents and market prices of the metals cannot be used as a basis of calculation of ore valuation until it is fully and definitely known where these metals will be recovered and what prices will be received for the various products.

Denver, Col. A. M. Plumb.

The Iron and Steel Market

The steel market turned quiet in the second half of December, but without losing a particle of its fundamental strength, the quietness being attributed to the usual holiday influence. Specifications on contracts expiring at the close of the month were as heavy as formerly, except for the influence that some buyers had already specified their limits. Most of the contracts, of course, were at relatively low prices.

On Dec. 15 the Carnegie Steel Company advanced its minimum price on bars, plates and shapes from 1.70c. to 1.80c., and as some of the other mills had already made such an advance the market became quotable at 1.80c. as minimum. Effective Dec. 21, the American Steel & Wire Company advanced wire prices $2 a ton, other mills immediately concurring.

The steel mills enter the new year with a very large volume of specific business on books, actual orders which were taken with the understanding that they were not to be canceled or the price revised no matter what occurred in the market. There is also a large volume of contract business upon which heavy specifications can reasonably be expected. The steel producers feel confident that, barring actual accidents in the market, they will be called upon to operate at their fullest capacity throughout the year 1916. All are careful, however, to avoid making the semblance of a prediction as to what may occur when the war ends. Now that a much smaller proportion than formerly of the total steel output is passing into war material, the question of the possible cancellation of war orders is relatively insignificant. The great question is the general economic changes that may be wrought, affecting the various industries of the United States. There is also the question of foreign competition, not only in neutral markets but possibly even in the markets of the United States. There will be a feverish revival of commercial activity in the belligerent countries, but there is a wide range of opinion as to whether this will be on a basis of high costs or low costs. All eyes will be turned to the United States as the place where the money lies. If the production of steel abroad is resumed more rapidly than the consumption there is the possibility of the offering of such steel greatly disturbing the markets in the United States.

Production of pig iron and steel has not been materially increased since October, a rate of substantially full production having then been reached. A few additional units have become productive, and others have been speeded up somewhat, but the weather, of course, has been less favorable than in September and October, and it may be said that production has been at an unchanged rate, with no prospects of material change.

Pig Iron

The various pig-iron markets have continued their advances, though as formerly without the stimulus, apparently, of any really large buying. Unless the advances represent an unusual amount of courage on the part of the furnacemen, an unprecedented hopefulness for the future, one must conclude that the trade is

facing a definite scarcity of pig iron, as there has been for four months a scarcity of steel. Pig iron has now advanced, since the low level maintained so consistently during the late months of 1914 and the first six months of 1915, fully $5 a ton. Such an advance is quite unusual. A moderate increase in demand serves to advance prices moderately, say, by from $1 to $4 a ton, merely through the operation of an economic law that as demand increases the less fit furnaces, with higher production costs, must be brought into operation. The market is now quotable as follows: No. 2 foundry iron, delivered Philadelphia, $19.25 to $19.75; at Buffalo furnaces, $18; delivered Cleveland, $18.80; at Chicago furnaces, $18 to $18.25; at furnaces, Birmingham district, $14 to $15; f.o.b. valley furnaces, 95c. higher delivered Pittsburgh: Bessemer, $19.50 to $20; basic, $18 to $18.50; foundry, $18.50 to $19; malleable, $18 to $18.50; forge, $18 to $18.50.

Steel

Since the scarcity of steel became acute there has been no regular open market for soft steel billets and sheet bars. The regular consumers were covered fairly well by long-term contracts, generally subject to a periodic adjustment of price, and the mills would not sell in the open market, or at least at such prices as would be paid by any considerable number of possible buyers. The pressure for billets has been accumulating, however, and some purchases of open-hearth are being made. One sale of about 2500 tons of soft open-hearth billets was made after the middle of December at $35, maker's mill, Pittsburgh district, and that may perhaps be regarded as fixing a market. Bessemer billets are quotable at $32 to $33, with but little tonnage available at that range. There is no demand for sheet bars at such prices, since prices obtainable for sheets and tin plate would not justify, and the consumers simply depend upon such steel as they obtain on their regular contracts. Rods are about $40 and forging billets about $50 to $55.

Finished Steel

Premiums for prompt shipment of plates are the rule, 2.25c. to 2.50c. being paid for small lots. Premiums are also occasionally paid for steel bars. Regular mill prices, f.o.b. Pittsburgh unless otherwise noted, are as follows:

Rails, standard sections, 1.25c. for Bessemer; 1.34c. for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.80c.

Shapes, 1.80c.

Steel bars and bands, 1.80c., base; steel hoops, 2c., base.

Iron bars, 2.009c. to 2.059c., Philadelphia; 1.75c., Chicago; 1.85c. to 1.90c., Pittsburgh.

Plain wire, 1.95c., base; galvanized wire, 2.65c.; wire nails, $2.10 per keg, base; painted barb wire, 2.25c.; galvanized barb wire, 2.95c.

Sheets, blue annealed, 10-gage, 2.25c. to 2.75c.; black, 28-gage, 2.60c. to 2.70c.; galvanized, 28-gage, 4.75c. to 5c.; painted corrugated, 28-gage, 2.80c.; galvanized corrugated, 28-gage, 4.80c. to 5.05c.

Tin plate, $3.60 for 100-lb. cokes.

Steel pipe, ¾ to 3-in., black, 78 per cent off list; galvanized 73½ per cent off list.

Boiler tubes (less than carloads), 68 per cent off list.

Structural rivets, 2.50c.; boiler rivets, 2.60c.

Cold-rolled shafting, 50 per cent off list.

Cold-rolled strip steel, 3.50c., base.

Non-Ferrous Metal Market

Dec. 27.—Unusual existing conditions are reflected in the metal market which is about as uncertain as the weather. Under normal conditions the last weeks of the year would be dull. The present market made a good start in this direction early in December, but the dullness was not sustained and heavy sales have been made during the last two weeks, especially in copper, forcing this metal to a very high level.

Copper.—The price has been held at close to 20 cents all during December up to last week, when large sales forced the price to 21½ cents. On the 22nd, 60,000 tons was contracted for by England for 1916 delivery, at a price said to be 21 cents delivered. Producers control the market and the metal is liable to remain in a very strong position for some time to come. Exports for November were 23,168 tons of 2240 lb., while exports for December are running considerably ahead of November.

Tin.—The price of tin has moved within a comparatively narrow range in December, buyers refusing to become excited. Doubtless they are tired of being misled by the many wild rumors. An absence of inquiry characterized the market early in December. On the 16th England announced the curtailing of exports, which, while it excited the market some, only caused a 1¼-cent advance, and toward the latter part of the month a reaction took place. Prices varied between 37.25 and 40.00 for spot during the month.

Lead.—The Trust price was raised by 0.15 to 5.40 on Dec. 14. The market is in a strong position and the advances recently made were expected. December exports are running ahead of November, which were 5423 tons.

Spelter.—The decline of 3 cents per pound which took place in a few days around Dec. 1 lowered the price to about 15 cents. Sellers became excited and attempted to unload their stocks. By the middle of the month sellers were refusing to quote, which shows how erratic the market is, or how erratic it is being made. For several days sellers refused to quote, consequently the market became excited and the price advanced to about 17.50 cents spot with slight variations up and down. At the time of writing spot spelter was 17.35-17.50.

Other Metals.—Aluminium held up well until Dec. 20, when the extreme dullness caused a decline, which forced the metal to 53.00-55.00. On the 27th it was quoted at 54.00-56.00. Antimony remained practically unchanged during December at 39.00-39.50 for both American, Chinese and Japanese. Silver was up to 56¼ on Dec. 6, since when it has gradually dropped to 53⅞. Tungsten continues at about $60 a unit for concentrates, platinum prices are nominal at $88 to $100 per ounce. Quicksilver is quoted at $125-$130 per flask.

One Hundred and Twelfth Meeting of American Institute of Mining Engineers

The 112th meeting of the American Institute of Mining Engineers will be held in New York City, Monday, Feb. 14, to Thursday, Feb. 17, inclusive. The committee on arrangements consists of David H. Browne, chairman; Bradley Stoughton, vice-chairman; Lawrence Addicks, P. E. Barbour, G. D. Barron, J. V. N. Dorr, J. R. Finlay, L. D. Huntoon, Burr A. Robinson, E. M. Shipp, Joseph Struthers, E. B. Sturgis and R. H. Vail.

A smoker and entertainment will be held on Tuesday evening, Feb. 15, which the committee in charge promises will be well worth attending. The annual dinner will be held at the Hotel Astor on Wednesday evening, Feb. 16, and the presence of ladies is urged. The dinner will be followed by dancing.

The entertainment of the ladies will be in charge of

the Ladies' Committee, of which Mrs. Louis D. Huntoon is chairman. A very complete program of entertainment has been provided for the ladies, including luncheons, teas, dances, theatres and a visit to the art galleries of Senator Wm. A. Clark. The ladies are also especially urged to enjoy the all-day boat trip which has been arranged for Thursday. By courtesy of the Secretary of the Navy a naval vessel will be provided for a cruise around the harbor. The firing of some of the large guns will be witnessed.

Monday evening, Feb. 14, will be "college night." A Committee on Reunions, of which Dr. Joseph Struthers is chairman, will be in charge. It is expected that many formal gatherings of college alumni will be held.

Many clubs kindly offered the extension of privileges to members last year, and it is expected they will do the same this year. All the facilities of the United Engineering Building will be at the service of the members and their guests, and from the looks of the preparations which are being made a very interesting and comfortable time should be had by all those attending.

The technical program will contain many interesting papers in the various sections on the different metals, petroleum and gas, mining and milling methods, coal, coke, geology, etc.

Meeting of the Naval Consulting Board

The U. S. Naval Consulting Board was in session on Dec. 22 and 23. The first day was mainly devoted to meetings of the different sub-committees, where general pending subjects were discussed and further steps were proposed for future work. On the evening of Dec. 22 the members were the guests at a short informal monthly dinner of the American Institute of Mining Engineers and then promptly left for the general meeting of the board.

The next day was devoted to an inspection of the submarines and battleships now assembled at the Brooklyn Navy Yard, after which another general meeting was held. At the meeting, the laboratory committee, consisting of Edison, chairman, and Baekeland, Whitney and Woodward as members, presented its final report and recommendations, which were accepted with much enthusiasm by the other members of the board as well as by the admirals who were present at the discussion. The laboratory provides for adequate facilities to make all tests and experiments relative to the construction and operation of men-of-war or ordinance and ammunition without the present delays which hamper so much the construction work of anything new and which compel postponement of trials of things until it is too late to make a change. In this respect alone the proposed laboratory would introduce an enormous saving of expenses aside from preventing the present unavoidable delays in the construction of our means of defense. Some time ago a firm which was more than one year behind in its deliveries, simply answered that the construction work involved a set of new devices, each of which involved serious problems which delayed the work, because the necessary data had to be gathered *during* construction work, which increased enormously the cost as well as the period of construction.

Among other reports which were communicated, we should mention the recommendation by the sub-committee on ordnance and explosives, that the Government of the United States should erect at least one unit for the synthetic manufacture of nitrates or nitric acid, so as to get at least one plant, however small, in which to learn to produce the indispensable chemical for the manufacture of smokeless powder or modern explosives. This unit would permit to train a staff of competent engineers or chemists to acquire the necessary experience and to provide means so that in time of war immediately other units could be added in a relatively short time. The plans, drawings, and specifications would be prepared in advance for such an emergency, and the initial staff would be able to train additional men for the work.

As matters stand now, it would be of little use to be equipped with best guns and warships as long as an enemy could easily seize the nitrate fields of Chile or occupy the ports of shipment of the valuable material. At present this country can provide within itself practically all other indispensable materials for war, except nitrates. Private industry has not gone into the matter of synthetic nitric acid here in the United States, because it can find better employment for capital in other chemical industries as long as it is so cheap and easy to import natural Chile saltpeter.

The Government might be able to run the enterprise at a small profit, or even at some loss, and use the produced nitrate for its own smokeless powder and explosive plants, or sell it as fertilizer and charge the relatively small investment as part of its equipment for defense while at the same time keeping abreast with an industry which ultimately may have an immense importance as a supply of cheap fertilizer for agricultural purposes.

The United States can well afford in its policy of conservation to set apart for this purpose one of the large water powers under its control.

The committee on manufacturing and standardizing is busying itself with extending the rational methods of manufacturing now adopted by the automobile manufacturers of this country to the rapid manufacture of aeroplane motors, and also proposes means whereby men of the army or the navy who want to become proficient in aeronautics will have the benefit of the unusually well provided instruction classes of the associated automobile manufacturers of this country free of charge to the Government.

In further accord with the idea of mobilizing the industries of the country a definite plan is being developed whereby the whole United States is divided into sections, and in each section a census is to be taken of the different industrial establishments and their equipment, and, furthermore, all necessary information is to be furnished to the manufacturers so as to enable them to convert their plants at shortest notice in time of war to contribute with best efficiency to the manufacture of ammunition and general supplies.

We are told that some other matters were presented and discussed, which, however, are not considered advisable for publication.

All these entirely practical tendencies of the Naval Board contradict flatly any apprehension which the ill-informed might have as to the vagaries of what they supposed to be a board of "long-haired inventors"—long-headed would be a more suitable definition.

Program of Meeting for Presentation of Perkin Medal to Dr. Baekeland

The presentation of the Perkin Medal to Dr. L. H. Baekeland will take place on Jan. 21, at a meeting of the Society of Chemical Industry in joint session with the American Chemical Society and American Electrochemical Society. The meeting will be held in Rumford Hall, Chemists' Club, at 8 p. m. The program includes the presentation speech, which will be made by Dr. C. F. Chandler, and the speech of acceptance by Dr. Baekeland.

The Western Metallurgical Field

The Everson Flotation Patents

Renewed public interest in the contributions of Carrie J. Everson to the process of concentration by flotation has been aroused by investigations started by the Colorado Scientific Society with a view to ascertaining her whereabouts and the incidents that led to her inventions. The Everson patents, issued to Carrie J. Everson of Chicago, Ill., (348,157, Aug. 24, 1886) and to Charles B. Hebron and Carrie J. Everson of Denver, Col., (471,174, March 22, 1892) disclose not merely the essentials of flotation as practised to-day, but enter into refinements of the process that show an unusually complete investigation.

In one section of the specifications, for example, it is stated that: "The essential feature of the method which constitutes my invention, therefore, consists in commingling with pulverized ore a fat or an oil, either animal, mineral or vegetable, or a fatty constituent or acid of an animal or vegetable fat or oil, or any constituent of a mineral oil, together with an acid, either mineral or vegetable, or a soluble neutral or acid salt, for the purpose of effecting a union of the free metal or metallic portion of the ore with such admixed material, whereby the same may be retained in the subsequent separation of the quartz or other rock, therefrom by washing or other suitable means."

Among the ores on which the inventor had successfully operated, are listed those containing native gold and silver, cerargyrite, argentite, argentiferous galena and a variety of double sulphides of silver, with copper, arsenic, antimony and other base metals. Pyrite containing gold and silver, and tellurides of gold and silver also were treated. Among the oils used are specified petroleum, paraffin oil, tallow, lard, lard oil, red oil (impure oleic acid), cotton-seed oil, castor oil, sperm and linseed oils. The list of acids includes sulphuric, hydrochloric, nitric, phosphoric, acetic, oxalic, tannic and gallic. The following salts also were used: sulphates and chlorides of sodium, zinc and copper and the double sulphate of potash and alumina.

Here is an unusually broad disclosure of factors which subsequent experience has proved essential to success, either when used singly or in combination. The ores mentioned are such as are treated to-day; the oils are similar, and, of course, the acids are practically the same as now used when deemed necessary. The mention of certain salts, such as sodium, zinc and copper sulphates and chlorides, is significant of the extent to which the research was carried, for it has been observed by later investigators that salts in solution are almost essential to good work. Practically pure water has been found a poor medium in which to work, and the addition of copper sulphate has, in a number of cases, proved the saving factor.

The investigation of the Colorado Scientific Society into the early history of the Everson patents was undertaken with a view to commemorating what now appears to have been a remarkable piece of research on the part of a Colorado woman; to establish the historical facts with accuracy and to prove or disprove, if possible, the legendary reports of the inception of the process. Up to date the following facts have been established with certainty: that Mrs. Carrie J. Everson was the wife of a physician, and for many years a resident of Denver, where she engaged in charitable work as a nurse. She was not a Denver school teacher and had no brother who was a Denver assayer.

The last few years of her life were spent in California, where she died at the age of 73, Nov. 3, 1914, at San Anselmo.

New Smelter Construction

Since 1907 the Coeur d'Alene district of Idaho has been second to Missouri in lead production in the United States; but in spite of the large quantity of ore and concentrate produced, none of the output has been smelted locally. The product has been shipped to smelters in Colorado, Utah and other States for reduction. It now appears that the Bunker Hill & Sullivan Mining & Concentrating Company is to erect a smelter in the Coeur d'Alene district, possibly near Kellogg, for the treatment of its own output and that of other mines in the district. The definite location of the plant has not yet been settled, but plans are in the hands of Jules LaBarthe, of Bradley, Bruff & LaBarthe, of San Francisco, and it is expected that definite announcement of location will soon be made. The establishment of this smelter is expected to prove of great value to the lead-mining industry of Idaho, and the outcome of the venture will be watched with interest.

Other Coeur d'Alene companies have interested themselves in the purchase and rehabilitation of the smelter at Northport, Wash., and this plant is expected to be in operation early in 1916. The Tamarack & Custer and Hercules companies are represented in the new Northport Smelting & Refining Company, of which J. J. Day is president, E. R. Day secretary-treasurer, and Edward Boyce, E. R. Knight and F. M. Rothrock directors. The plant is to be equipped for both lead and copper smelting. Lead ores will be received from the mining companies directly interested in the smelter, and custom ores from other companies in the Coeur d'Alenes. A supply of copper ore is now being arranged for in the Chewelah district.

Fields Flotation Process at Ohio Copper Mill

It is announced that the Fields flotation process is to be tried experimentally at the mill of the Ohio Copper Company, near Salt Lake City, Utah. Some months ago arrangements were made for a test of this process at the Ohio mill, but the company became involved in financial difficulties which deferred action in the matter. The process is one developed by John D. Fields at the Royal Basin property, Maxville, Mont., and is conducted in an electrolytic cell. Electrolysis is said to produce two results: the generation of gas to buoy the oiled sulphide minerals, and the deposition of copper dissolved by the action of acid on oxidized minerals which may be present in the ore. The work of installing the 25-ton experimental unit is in charge of Lee M. Walton, representing the Fields company.

Activity in the Joplin District

In no section of the country have the high prices for zinc and lead had a more marked effect than in the Joplin district of Missouri. Early in November the valuation of zinc and lead produced in 1915 was in excess of $20,000,000, or more than $2,000,000 greater than for 1912, the previous banner year. With the output for November and December to be added to the production to date, the total valuation for 1915 will approximate $24,000,000. The production of blende is forecasted at 250,000 tons, calamine 18,000 and galena 42,000. For the week ended Nov. 14, the base price paid for blende concentrate containing 60 per cent zinc ranged from $85 to $110. For the corresponding week of 1914 the range was $40 to $44. Calamine has been in strong demand at $65 to $75, basis of 40 per cent zinc, as compared with $19 to $21 a year ago. Galena ranged from $60 to $64, basis of 80 per cent lead, compared with $42 at a corresponding period in 1914.

These conditions have stimulated mill construction which has proceeded at an unprecedented rate, both for

the treatment of old tailings and ore from new development. In the Carterville district a 300-ton mill has been built on the Old Century and Herald mines; a 150-ton tailing mill on the land of the South Carterville Mining Company; a 300-ton mill by the Ben Franklin company; a 250-ton mill for the Queen Esther company. The mill of the Twin Cities company has been remodeled.

At Galena a 750-ton tailing mill has been built on the Squires land; and the Grand Tower Mining Company has given a contract for the construction of a 150-ton mill on the Campbell land.

The Kusa smelter is a new plant in Oklahoma which will treat Joplin ore.

Arizona Copper Company's Fire Loss

The troubles of the Arizona Copper Company, due to labor difficulties since September, were augmented on the fifth of November by the destruction by fire of parts of the concentrating plant at Clifton. The loss is estimated variously at from a quarter to a half million dollars. As soon as the fire was discovered in No. 2 concentrator it was fought by the local fire department and men on strike. Before it was controlled the flames had destroyed No. 3 department also, and was checked only by the use of dynamite. Union sympathizers state that the fire was due to defective wiring, but this is not accepted by the company officials. It is reported that new plants will be erected as soon as labor troubles are settled.

Company Reports

In the annual report of the **Nevada Wonder Mining Company** is contained the résumé of operations by the **Churchill Milling Company**, which treats Nevada Wonder ore. The mill made a record during the year ended Sept. 30, 1915, treating 58,124 tons of ore, an average of 159.2 tons per day, and an increase of 26.1 tons per day over the preceding year. Based on actual running time, the stamps showed a tonnage of 18.01 per stamp per twenty-four hours. Gold recovery was 95.3 per cent and silver 93.68 per cent. Direct milling cost was $2.268 per ton; indirect, $0.545; total cost, $2.813 per ton. Cost of power per ton was $0.394; supplies, $1.166; labor, $0.705.

The Granby Cons. Mining, Smelting & Power Company, Ltd., smelted 1,098,020 dry tons of ore at its Grand Forks and Anyox plants in British Columbia during the year ended June 30, 1915. Of this amount only 24,583 tons was foreign ore. The production was 26,638,912 lb. refined copper, 377,881 oz. fine silver and 31,388 oz. fine gold. The cost of Granby copper was 10.66 cents per pound. Conditions at the new Anyox plant have improved as operations have proceeded. New charge cars have modified the feeding and distribution of the ore in the furnaces so that less trouble is experienced from crusts in the shafts. A new furnace, No. 4, and an agglomerating plant have been added to the equipment. Due to the erratic silica content of the ores received it has been impossible to operate on the semi-pyritic basis as contemplated, with low coke and little flux; but with the development of the company's own ore bodies it is expected that conditions can be more closely controlled. It is expected that the making of converter grade of matte in the first smelting will finally be accomplished. At Grand Forks the plant was closed from Aug. 7 to Dec. 7, 1914, and has since been put in full operation. Operations at this plant are on a well established basis and few difficulties are encountered. The cost of smelting was $1.187 per ton, as against $1.214 for 1913.

The average percentage of coke used per ton of ore was 13.17.

Baltimore Meeting of American Institute of Chemical Engineers

The eighth annual meeting of the American Institute of Chemical Engineers will be held in Baltimore, Md., from Wednesday to Saturday, Jan. 12 to 15, 1916.

On Wednesday, Jan. 12, the annual business meeting will be held at 10 a. m., at the Emerson Hotel, followed at 11 a. m. by the reading and discussion of the following two papers:

The Development of the Manufacture in the United States of Products Derived from Coal, by H. W. Jordan.

The Cracking of Petroleum and Other Hydrocarbons as a Chemical Engineering Problem, by W. F. Rittman.

In the evening of Wednesday, at 8 p. m., a meeting will be held at Johns Hopkins University, when the following three papers will be presented:

Ethyl Alcohol from Wood Waste, by A. D. Little.

The Production of Ammonia from Cyanamid, by W. S. Landis.

The Barium Industry, by M. Toch.

On Friday, Jan. 14, after a business meeting, at 9 a. m., a professional meeting will be held at 10 a. m., at the Emerson Hotel for the reading and discussion of the following papers:

A New Process of Bleaching, by S. F. Peckham.

Lutes and Cements for Chemical Purposes, by S. S. Sadtler.

On the evening of Friday, a joint meeting will be held with the Baltimore Section of the American Chemical Society at Johns Hopkins University, when Mr. Powell, of the Baltimore County Water Co., will discuss ozone purification of water and Dr. H. E. Ives will read a paper on artificial daylight.

On the evening of Thursday, a subscription dinner will be held at 8 p. m., at the Emerson Hotel. On the evening of Friday, a smoker will be tendered at the Johns Hopkins Club by the Maryland Section of the American Chemical Society.

The following excursions have been arranged:

For Wednesday afternoon, to the sewage disposal plant of the City of Baltimore and the ozone water purification plant of the Baltimore County Water Co., or to the plant of the Maryland Steel Co. at Sparrow's Point.

For Thursday (all day), to Emerson Drug Co. (bromo seltzer manufacturers) and to the Naval Academy in Annapolis.

For Friday afternoon, to the Maryland Glass Company and Crown Cork and Seal Company or to the Standard Fertilizer Works.

For Saturday, to McCall's Ferry Power Plant of the Pennsylvania Water & Power Co. or to the Tidewater Portland Cement Co., at Union Bridge, Md.

Members will be welcome at the chemical and physical laboratories of Johns Hopkins University any day during the meeting between 9 a. m. and 5 p. m.

The local committee consists of Messrs. Richard K. Meade, J. G. Dailey and W. T. Van Horn, while Dr. E. Emmet Reid represents the Maryland Section of the American Chemical Society on the committees.

Prof. J. C. Olsen, Cooper Union, New York City, is secretary of the American Institute of Chemical Engineers.

Coal in the United States.—The recently issued annual report of the Geological Survey for 1914 on the production of coal contains an interesting chart of the yearly production of coal from 1807, the date of the first discovery, to 1914.

Thermal Reactions in the Vapor Phase of Various Coal Tar Oils and Distillates*

BY WALTER F. RITTMAN AND GUSTAV EGLOFF

Previous experiments of the present general series have indicated clearly the direction and mechanism of thermal reaction among hydrocarbons. It has been shown[1] that paraffin hydrocarbons tend to form: (1) other paraffin hydrocarbons of smaller molecular weight, (2) unsaturated aliphatic hydrocarbons, and (3) aromatic hydrocarbons of mono- and poly-cyclic series. It has been shown also that inter-aromatic reactions may be arranged in a definite and consistent series. Mono-cyclic compounds with aliphatic side chains tend to form benzene, and benzene is decomposed by heat into naphthalene, anthracene and other poly-cyclic compounds.

A study[2] of the relative content of aromatic hydrocarbons of cracked petroleum mixtures led to conclusions of the same sort. It appeared that toluene and xylene were the products of less strenuous conditions of cracking than benzene, and it appeared also that naphthalene and anthracene were the results of the decomposition of the mono-cyclic compounds. In all the work it was indicated in a definite manner that the reactions are practically irreversible. For instance, no appreciable quantity of benzene is formed by the heating of naphthalene.

SCOPE OF THE PRESENT WORK

The present experiments were conducted for the purpose of testing the utility of various low-grade coal tar oils and distillates as original material for the production of the valuable hydrocarbons, toluene and benzene. On the basis of indications furnished by the results of previous work, it appears that transformations may be effected advantageously with such mixtures as contain mono-cyclic compounds with aliphatic side chains. The higher boiling distillates of the light oil cut, containing such hydrocarbons as xylenes, ethyl benzene, mesitylene, and cymene, would appear the most promising material possible, since the change involves only the simple splitting of a side chain attached to an exceedingly stable cyclic nucleus. On the other hand, products which consist chiefly of compounds with poly-cyclic nuclei would not seem to be utilizable, since the results of all previous experiments have indicated that the simpler hydrocarbons cannot be produced by the breaking down of the more complex aromatic compounds.

EXPERIMENTAL

The method of procedure was, in general, the same as that which has been adopted for all the series of experiments on vapor phase reactions of hydrocarbons. Details of operation were identical with those of the experiments[3] in which pure aromatics were used as original materials. The oils were run through an electrically heated cracking furnace, and the resulting products collected and studied. The yields of solid, liquid and gaseous products were measured and the liquids analyzed. The latter were distilled with Hempel fractionating columns and cuts made at 75 deg., 100 deg., 125 deg., 150 deg., and thereafter at intervals of 50 deg. Specific gravities of the recovered oils and of the distillation cuts were measured. The fractions coming over at higher temperatures were examined for the presence of naphthalene and anthracene by the simple method of chilling and noting the appearance of solid matter frozen out.

OILS USED AS ORIGINAL MATERIAL

The oils used as original material were all coal-derived products and were either of medium or of low cost. Their properties are indicated by the following descriptions:

Solvent Naphtha was a hydrocarbon mixture of slightly higher boiling point than the benzene and toluene fractions of coal tar and probably consisted chiefly of xylenes and cymene. It usually sells for a low price and is used as a solvent; but is now in such demand that it is moderately expensive, although still much cheaper than benzene and toluene. The sample used in the present experiments was obtained from a large company specializing in the working up of coal-tar products.

Creosote Oil was the ordinary product sold under that name. The sample employed in the present experiments was obtained from a second company handling coal-tar products.

Light Dead Oil was one of the typical low grade coal-tar fractions. It was obtained from the second company mentioned.

Naphthalene Wash Oil was the drip oil which comes from the presses used in separating naphthalene. It was obtained from the company furnishing the two preceding oils.

Imperial Varnish Oil was similar in properties to the solvent naphtha but was slightly heavier than the product usually sold under the latter name. The sample was obtained from a company operating in Canada.

Heavy Dead Oil was slightly heavier than the light dead oil, but contained a larger percentage of constituents boiling below 200 deg. C. It was obtained from the company furnishing the solvent naphtha.

Light Tar Oil was an intermediate product between the solvent naphtha and the cresote oil. It was similar in properties to the light dead oil.

Secondary Oil Residue was a product obtained from a steel company which operates its own by-product coke ovens. The oil was the residue left after a distillation of the product obtained by the primary fractionation of the saturated wash oil coming from the gas scrubbers.

GENERAL RESULTS OF EXPERIMENTS

Results of the present series of experiments appear in Table I. It will be noted that the sets of experiments performed for the various oils were irregular. This is accounted for by the fact that it was not considered necessary to study systematically the effects of temperature and pressure. In several cases the number of experiments performed with a single oil was limited by the size of the sample. In most cases conditions of temperature and pressure were selected which had been indicated by previous general experience as most favorable for cracking the oils in question. A few runs sufficed to yield positive results with some products, while with others, when results were of a negative nature, the ranges of temperature and pressure were more extensive. In all cases the results obtained were entirely in accord with what was expected from theoretical considerations and from the indications of previous experience.

The figures in Table I represent the specific gravities, percentage recoveries, and quantities of carbon formed, as well as the composition of the recovered oils. Distillation cut figures are given in terms of the original oil rather than on the basis of recovered oil. This in all cases reduces the apparent magnitude of the yields of benzene and toluene.

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*Published with the permission of the Director of the Bureau of Mines.

[1] Rittman, Byron, and Egloff—Jour. of Ind. and Eng. Chem., 7 (Dec., 1915), 1019.

[2] Results about to be published.

[3] Rittman, Byron, and Egloff, loc. cit.

TABLE I—RESULTS OF CRACKING VARIOUS COAL TAR OILS AND DISTILLATES BY THE VAPOR PHASE METHOD. FIGURES SHOWING PERCENTAGE RECOVERIES, SPECIFIC GRAVITIES AND DISTILLATION CUTS OF LIQUID PRODUCTS, AND PERCENTAGES OF CARBON FORMATION

Oil	Solvent Naphtha								Creosote Oil							
Temp., deg. C.	Original	600	650	700	600	650	550		Original	550	600	650	600	650		
Pressure, atmosphere	Oil	1	1	1	8	8	13		Oil	1	1	1	11	11		
Per cent recovered	.....	78.3	32.5	31.7	70.8	35.8	73.3		.....	93.3	80.0	50.0	75.0	37.5		
Specific gravity	.....	.....	.....	.....	.....	.....	.....		1.095	1.095	1.090	1.093	1.053	1.060		
Per cent carbon	.....	Tar	0.5	3.5	2.6	18.0	1.9		.....	0.3	0.5	1.0	1.8	3.0		
Distillation cut (on basis of original oil:)																
75 deg.-100 deg.	1.2 .868	4.5 .865	3.1 .868	7.7 .876	9.9 .874	12.5 .867	2.3 .862		—	0.5 —	0.8 .866	0.6 .867	2.6 .840	1.1 .844		
100 deg.-125 deg.	1.5 .868	7.8 .871	13.9 .875	5.2 .868	19.5 .846	5.7 .869	9.5 .868		—	1.2 .864	1.3 .868	0.5 .869	1.9 .850	0.9 .860		
125 deg.-150 deg.	70.5 .864	44.6 .874	21.0 .873	5.8 .866	23.0 .845	10.0 .877	44.7 .869		—	1.6 .866	1.5 .869	0.4 .869	2.3 .855	1.3 .878		
150 deg.-200 deg.	6.7 .864	.....	.....	.....	.....	.....	.....		2.5	.....	.....	.....	.....	.....		
Napthalene	—	+	+	+	+	.....	.....		+	+	+	+	+	+		
Anthracene	—	+	+	+	+	.....	.....		+	+	+	+	+	+		
Oil	Light Dead Oil															
Temp., deg. C.	Original	600	650	700	650	700	600		700	600	650	750				
Pressure, atmosphere	Oil	1	1	1	8	8	11		11	15	15	15				
Per cent recovered	.....	87.0	86.0	62.0	63.1	48.0	70.3		35.0	67.0	43.0	33.0				
Specific gravity	1.007	1.022	1.048	1.060	1.040	1.028	1.015		1.028	1.014	1.040	1.020				
Per cent carbon	.....	.....	1.0	2.0	20.0	25.0	7.6		26.0	10.0	24.0	25.2				
Distillation cut (on basis of original oil):																
75 deg.-100 deg.	—	0.9 —	1.5 —	0.5 —	2.5 .875	0.9 .874	2.1 .879		1.0 .878	1.5 .848	0.9 .878	2.7 .879				
100 deg.-125 deg.	—	1.0 .876	1.5 —	1.1 .852	1.3 .875	0.9 —	1.0 .880		0.7 .884	1.3 —	0.9 .880	1.7 .878				
125 deg.-150 deg.	4.0 .860	2.6 —	3.3 —	2.2 .878	1.3 —	1.9 —	1.4 .885		0.9 .883	2.7 .841	1.3 —	1.3 .876				
150 deg.-200 deg.	5.5 .945	.....	.....	.....	.....	.....	.....		.....	.....	.....	.....				
Napthalene	+	+	+	+	+	+	+		+	+	+	.....				
Anthracene	—	+	+	+	+	+	+		+	+	+	.....				
Oil	Naphthalene Wash Oil										Imperial Varnish Oil					
Temp., deg. C.	Original	550	600	650	550	600	650		550	600	Original	700	650	700		
Pressure, atmosphere	Oil	1	1	1	8	8	8		11	11	Oil	1	8	8		
Per cent recovered	.....	81.7	71.7	40.0	70.8	51.7	29.2		45.8	42.0	.....	70.8	66.7	33.3		
Specific gravity	.....	988	1.010	1.025	1.045	1.036	1.022		1.028	1.032	929	926	.965	970		
Per cent carbon	.....	.....	.....	3.3	1.3	8.8	26.8		2.0	14.9	.....	.....	1.3	10.0		
Distillation cut (on basis of original oil):																
75 deg.-100 deg.	.....	1.6 .822	2.1 .856	1.8 .870	1.8 .844	4.1 .868	2.8 .874		3.2 .858	4.0 .875	—	2.1 —	6.8 .877	12.6 .877		
100 deg.-125 deg.	0.5 .878	1.9 .850	1.9 .868	1.6 .869	3.0 .874	3.3 .870	1.2 .870		2.1 .866	2.5 .870	4.5 .865	11.3 .856	10.7 .875	2.6 .875		
125 deg.-150 deg.	2.2 .878	4.2 .878	5.6 .875	5.1 .878	5.9 .880	2.9 .875	2.9 .874		3.2 .872	2.2 .876	22.0 .875	13.5 .863	8.8 .879	4.7 .880		
150 deg.-200 deg.	23.0 .947	.....	.....	.....	.....	.....	.....		.....	.....	38.0 .888	.....	.....	.....		
Napthalene	+	+	+	+	+	+	+		+	+	+	+	+	+		
Anthracene	.....	+	+	+	+	+	.....		+	+	—	.....	+	—		
Oil	Heavy Dead Oil			Light Tar Oil							Secondary Oil Residue					
Temp., deg. C.	Original	650	700	Original	550	600	650		600	650	Original	650	700	650		
Pressure, atmosphere	Oil	1	1	Oil	1	1	1		11	11	Oil	1	1	11		
Per cent recovered	.....	66.0	64.0	.....	95.0	87.5	82.5		75.8	47.5	.....	74.0	58.0	66.7		
Specific gravity	1.010	1.033	1.045	.984	.967	.996	1.013		.983	.992	.979	.983	1.050	.981		
Per cent carbon	.....	Tar	Tar	.....	0.0	0.0	0.0		2.0	10.0	.....	Tar	Tar	25.0		
Distillation cut (on basis of original oil):																
75 deg.-100 deg.	1.6 .889	2.8 .876	2.5 .879	6.7 .857	6.7 .849	7.6 .869	10.3 .867		6.1 .879	9.5 .883	—	1.1	2.2 .873	5.7 .861		
100 deg.-125 deg.	0.3 —	1.3 .882	1.0 .883	7.5 .865	10.3 .864	10.2 .875	8.2 .873		7.9 .878	2.5 .881	....	0.7 .801	3.5 .876	2.2 .868		
125 deg.-150 deg.	0.5 —	3.3 .900	2.9 .903	10.3 .873	7.8 .876	6.2 .881	6.8 .882		6.1 .886	1.9 .887	1.6 .880	3.0	5.5 .901	5.2 .879		
150 deg.-200 deg.	28.0 .983	.....	.....	27.3 .955	.....	.....	.....		.....	.....	47.8 .958	.....	.....	.....		
Napthalene	....	+	+	+	+	+	+		.....	+	+	.....	+	+		
Anthracene	....	+	+	—	—	.....	+		.....	+	.....	.....	.....	+		

TABLE II—SECTION A. PERCENTAGE RECOVERIES OF LIQUID REACTION PRODUCTS WHEN COAL-TAR OILS ARE CRACKED UNDER VARIOUS CONDITIONS BY THE VAPOR PHASE METHOD

Temp., deg. C.	550	600	650	700	550	600	650	700	550	600	650	700	600	650	700
Pressure, atmosphere	1	1	1	1	8	8	8	8	11	11	11	10	15	15	15
Solvent naphtha	.....	78.3	52.5	21.7	.....	70.8	35.8	.....	.....	.....	.....	.....	.....	.....	.....
Creosote oil	93.3	80.0	50.0	.....	.....	.....	.....	.....	.....	75.0	37.5	.....	.....	.....	.....
Light dead oil	.....	87.0	86.0	62.0	.....	.....	63.1	48.0	.....	70.3	.....	35.0	67.0	43.0	33.0
Naphthalene wash oil	81.7	71.7	40.0	.....	70.8	51.7	29.2	.....	45.8	42.0	.....	.....	.....	.....	.....
Imperial varnish oil	.....	.....	.....	70.8	.....	.....	66.7	33.3	.....	.....	.....	.....	.....	.....	.....
Heavy dead oil	.....	.....	66.0	64.0	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Light tar oil	95.0	87.5	82.5	.....	.....	.....	.....	.....	.....	75.8	47.5	.....	.....	.....	.....
Secondary oil residue	.....	.....	74.0	58.0	.....	.....	.....	.....	.....	66.7	.....	.....	.....	.....	.....

SECTION B. PERCENTAGE RECOVERIES WHEN PETROLEUM IS CRACKED BY THE VAPOR PHASE METHOD

Temp., deg. C	600	600	600	600
Pressure, atmosphere	1	6	12	18
Pennsylvania	59.9	31.4	43.6	40.5
California	46.6	43.9	52.5	33.7

Distillation cuts are in terms of original oil, the values given being obtained by multiplying the actual distillation figure obtained by the percentage recovery.
The volume of original oil used in each run was 560 cc.
Pressures are given in approximate atmospheres. The actual pressures were atmospheric and 100 pounds, 150 pounds, 175 pounds, and 200 pounds above atmospheric. The latter four pressures are approximately 8, 11, 13, and 15 atmospheres.

MAGNITUDE OF PERCENTAGE RECOVERY

In Table II are given the figures for percentage recoveries, these being collected from Table I. The first point to be noted is that these values are considerably greater than those for recoveries from petroleum cracked under approximately similar conditions. A few figures are given, taken from the results of a previous set of experiments.[4] California and Pennsylvania oils were selected as representative of different classes. The observation that coal-tar oils are less extensively decomposed than petroleum is exactly what would be expected, since the former are the products of a primary thermal decomposition at high temperature.

It may be observed also that there are more or less characteristic variations among the different coal-tar oils, which, however, on account of the irregularity of the series it is not possible to indicate by averages. The general tendency is toward greater decomposition on the part of those oils which have larger percentages of mono-cyclic constituents (indicated by the magnitude and specific gravity of the cut distilling below 200 deg. C.).

[4] Rittman, Jour. of Ind. and Eng. Chem., 7 (1915), 945.

SPECIFIC GRAVITY OF RECOVERED OILS AND PERCENTAGES OF CARBON FORMATION

No points of particular interest are brought out by the figures for specific gravities of recovered oils and percentages of carbon formation. These values have not, therefore, been retabulated. The one observation which may be made is that neither the changes in specific gravity, due to cracking, nor the percentages of carbon formed, are as great as in cases when petroleum was the original material. This is again in accord with what would be expected when consideration is given to the fact that the coal-tar oils are the products of a primary thermal decomposition at high temperature.

FORMATION OF BENZENE AND TOLUENE

The essential point concerning which information has been sought in the present experiments is the degree of formation of the commercially valuable hydrocarbons, benzene and toluene. Since the products are not contaminated with any considerable quantities of aliphatic hydrocarbons their degrees of formation may be represented by the magnitude of the respective distillation cuts, 75–100 deg. and 100–125 deg. In the present case the amount of original oil distilling up to 200 deg may, likewise, be taken as representative of its content of mono-cyclic aromatics. The figure 200 deg. is not strictly accurate and a temperature of 15 deg. to 20 deg. lower would be better; but, as no cuts were made at this point, it cannot be used as a limit of separation. It has seemed that there ought to be an approximate relation between the amounts of benzene and toluene formed and the percentage of mono-cyclic aromatics in the original oil. That such a relation exists is indicated by the figures collected in Table III.

The following values appear:

Column I.—Percentage of benzene cut, on basis of original oil.

Column II.—Percentage of toluene cut, on basis of original oil.

Column III.—Sum of percentages of benzene and toluene cuts.

Column IV.—Percentage of original oil distilling below 200 deg. C.

Column V.—Ratio representing the degree of transformation of the 200 deg. C. original cut into benzene and toluene. This figure is obtained by dividing the value in Column III by that in Column IV.

Column VI.—Conditions of temperature and pressure under which the oils were cracked in the runs for which figures are given.

The figures are in all cases for optimum runs; those being selected which gave maximum total yields of benzene and toluene. With one exception the figures in Column V are of the same order. They indicate that the quantity of benzene and toluene formed cannot exceed the content of the original oil of mono-cyclic hydrocarbons and that the degree of formation seems to have a maximum limit of about 50 per cent. The figures obtained would undoubtedly agree even more closely than they do if it were not for the necessary inaccuracies of assumption and of experimental manipulation. The case of the creosote oil is difficult to explain on the basis of any of the more probable causes of inaccuracy, but as it deals with a small 200-deg. cut and a small quantity of benzene and toluene formed it does not seem of great importance.

In general, it appears that by cracking under optimum conditions somewhere between 20 and 40 per cent of the portion of the original oil distilling below 200 deg. C. may be converted into benzene and toluene. The figure probably may be in extreme cases as low as 10 per cent or as high as 50 per cent.

While the indications of Table III are only qualitative and approximate, they are still of great value in determining the utility of coal tar distillates for transformation into benzene and toluene. Products like the creosote oil and the light dead oil are unquestionably useless as raw material for this purpose. Products like the solvent naphtha and varnish oil are the best possible material, although their use may be restricted on account of the factor of cost.

SUMMARY

Several coal-tar oils and distillates, all of the class less valuable commercially, have been subjected to cracking by the vapor phase method. Conditions of temperature and pressure have been chosen which seemed particularly adapted to the oils in question.

It appeared that percentage recoveries were high, as compared with those obtained when runs were made with petroleum as original material. Changes in specific gravity of liquid product and degrees of carbon formation are likewise lower than in the case of petroleum. These results were expected in consideration of the fact that the coal-tar products have once before been subjected to high temperature and brought at least approximately to equilibrium.

Oils and distillates having a considerable percentage of constituents distilling below 200 deg. C. yield large quantities of benzene and toluene, the yield being generally between 20 and 40 per cent of the 200 deg. cut from the original oil. Oils having minor percentages of constituents distilling below 200 deg. yield the same proportion of the desired hydrocarbons, but as this is likewise a small quantity they are undesirable material for transformation.

The failure to obtain benzene and toluene in such quantities as to indicate that hydrocarbons boiling above 200 deg. have been transformed is in accord with previously obtained results.[5]

These demonstrated that inter-aromatic reactions are irreversible, and that they proceed in the order indicated by the following series:

[5]Rittman, Byron, and Egloff, loc. cit. Rittman and Twomey, loc. cit.

TABLE III—BENZENE AND TOLUENE FORMATION, AS INFLUENCED BY CONTENT OF MONO-CYCLIC AROMATICS OF ORIGINAL OIL. RATIO BETWEEN AMOUNT DISTILLING UP TO 200 DEG. IN ORIGINAL OIL AND THE SUM OF THE BENZENE AND TOLUENE CUTS

	Column No. I	II	III	IV	V	VI Conditions of Cracking	
	Benzene	Toluene	Benzene and Toluene	200 Deg. Orig. Distillate	Ratio Between III and IV	Temp., Deg. C.	Pressure, Atm.
Solvent naphtha	9.9	19.5	29.4	79.9	.37	600	8
Creosote oil	2.6	1.9	4.5	2.5	1.8	600	11
Light dead oil	2.7	1.7	4.4	9.5	.46	700	15
Naphthalene wash oil	4.1	3.3	7.4	25.7	.29	600	8
Imperial varnish oil	6.8	10.7	17.5	64.5	.27	650	8
Heavy dead oil	2.8	1.3	4.1	30.4	.13	650	1
Light tar oil	10.3	8.2	18.5	51.8	.36	650	1
Secondary oil residue	5.7	2.2	7.9	49.4	.16	650	11

Higher benzene homologues, lower benzene homologues, benzene, naphthalene, anthracene, etc.

The only aromatic hydrocarbons capable of transformation into benzene and toluene are the higher benzene homologues, and these in general have boiling points of 175 deg. C. and below.

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Experiments on the Separation of Vanadium from Crude Sodium Uranate*

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In the extraction of radium from carnotite ore considerable quantities of uranium and vanadium compounds are obtained as by-products. When the uranium is separated by precipitation as sodium diuranate, $Na_2U_2O_7$, from solutions also containing compounds of vanadium a crude uranate of sodium is obtained, generally bearing from 5 to 10 per cent vanadium, expressed as the pentoxide, V_2O_5. Our experiments constitute a study of several laboratory methods for the separation of vanadium in the crude uranate, and the conversion of the uranate into uranium oxide.

Since sodium vanadate is very soluble in water and sodium uranate is not, we would expect that the vanadium could be completely separated from the vanadium bearing uranate by solution of the latter in one of the mineral acids, and precipitation of the uranium by the addition of sodium hydroxide in excess. This operation does remove about one-half of the vanadium, but the yellow uranate of sodium still contains too high a percentage of vanadium for market demands. To reduce the percentage of vanadium pentoxide to less than one-half of 1 per cent by this method would evidently require four to five repetitions. Our experiments were directed toward finding a more direct method of removing the vanadium.

The stock supply of crude uranate used in these experiments was furnished by the National Radium Institute through the courtesy of Mr. R. B. Moore, physical chemist of the U. S. Bureau of Mines, in charge of the radium investigations at Denver. Several pounds of this material were ground and sampled. An analysis was made by following, in the main, the directions given in Bulletin No. 70, U. S. Bureau of Mines, page 88, for the analysis of carnotite ores. The results of the analysis are summarized in Table I.

Table I—Analysis of Crude Uranate—Percentages

Constituent	Sample No. 1	Sample No. 2	Average
Water at 120 deg.	2.91	3.33	3.12
Water of constitution	7.03	6.40	6.72
Insoluble residue	0.28	0.31	0.29
Copper and tin group oxides	0.23	0.37	0.30
Calcium oxide, barium oxide and ferric oxide	5.96	6.04	6.00
Uranium oxide U_3O_8	64.26	65.00	64.63
Vanadium oxide V_2O_5	8.60	8.50	8.55
Sodium (metal)	7.90	8.33	8.12
Total	.....	.....	97.73

The apparently low total is due to the fact that for every three molecules of sodium diuranate, $Na_2U_2O_7$, there are five atoms of oxygen which are not accounted for in the above results. If we calculate the amount of oxygen to be added, assuming that the crude uranate contains 64.63 per cent of the oxide, U_3O_8, the per cent of oxygen not accounted for would amount to 3.07 and the total percentage then is 100.8.

Working on a laboratory scale, we have succeeded in reducing the vanadium content of the crude uranate to less than one-half of 1 per cent by several methods: (1) Removal of the vanadium as a volatile product by treatment with hydrochloric acid gas. (2) Treatment with ammonium chloride. (3) Separation of vanadium by treatment with some of the common mineral acids.

Treatment of the Crude Uranate with Hydrogen Chloride

Edgar F. Smith and Joseph B. Hibbs[1] first showed that vanadic acid can be completely removed from sodium vanadate as a volatile product by means of hydrochloric acid gas. Later, it was shown by Hillebrand that a part of the vanadium in ores, the vanadous compounds, are not completely removed by this method. Our first experiments were undertaken to ascertain whether the vanadium in the crude uranate can be volatilized by hydrogen chloride. The crude uranate was subjected to the action of hydrochloric acid under varying conditions of temperature. It was found that crude sodium uranate which had been ignited in the free flame to remove moisture present, and then subjected to dry hydrogen chloride evolves brown fumes characteristic of the chlorides and oxychlorides of vanadium. The reaction proceeds fully as well at room temperatures as when the uranate is heated. Evidently the reaction produces enough heat to raise the temperature sufficiently to volatilize the chlorides and oxychlorides of vanadium.

During the process a slight caking is produced as a result of the water formed in the reaction, and this sometimes prevents the hydrogen chloride from interacting with the vanadium in the interior of the uranate. In such cases it is necessary to heat the uranate residue to a temperature high enough to expel all the moisture present, then to grind it to a fine consistency. When hydrogen chloride is passed over it a second time a slight evolution of brown fumes is again observed, showing that only traces of the vanadium are left. Two treatments with hydrogen chloride gave a very complete quantitative separation of the vanadium.

The residue left after the separation of vanadium probably is a mixture of sodium chloride, sodium uranate, and uranyl chloride. This residue is readily soluble in dilute acids, from which the uranium can be obtained in the form of oxides by either precipitation with ammonia and ignition of the uranate, or else by boiling with ammonium chloride, which likewise converts the major part of the sodium uranate into ammonium uranate.

In subjecting the vanadium bearing uranate to the action of hydrogen chloride, the uranate was placed in a long, hard-glass tube of 1-cm. bore. One end of the tube was fitted with a one-hole stopper, carrying a glass tube. A steady stream of hydrogen chloride was passed over the uranate in the tube and maintained until the evolution of the brown fumes ceased. The reaction was carried out at different temperatures by placing the glass tubes in a combustion furnace and regulating the temperature.

In order to get some idea of the quantitative separation of the vanadium from the crude uranate by this method, the volatile compounds of vanadium from a weighed sample of uranate were absorbed in water. The aqueous solution of vanadium was oxidized with hydrogen peroxide, and evaporated nearly to dryness with sulphuric acid. It was taken up with water and reduced with sulphur dioxide. The solution was then boiled to expel the excess of sulphur dioxide, after which the hot solution was titrated with a standard solution of potassium permanganate. The amount of

*Extract of a thesis submitted by Howard H. Barker in the Graduate School of the University of Missouri in partial fulfillment for the degree of Master of Arts, June, 1915.

[1]Jour. Am. Chem. Soc. 16, 578 (1894).

vanadium in the aqueous solution was found to vary from 8 to 8.2 per cent of V_2O_5, as against 8.55, the amount actually found in the crude uranate by analysis. This result shows that the vanadium is nearly removed quantitatively. The residue showed only a feeble test for vanadium.

Whether this method can be conducted on a commercial scale is an open question. In designing a large form of apparatus, a chemical engineer must consider the following points: (1) The material to be used in the construction of the apparatus, wood, metal or stoneware. (2) The process should be made continuous. That is, all the hydrogen chloride should be utilized by passing that which is rich in the products of the reaction over fresh crude uranate. (3) The water formed during the reaction is sometimes very objectionable, as it produces caking. The apparatus should, therefore, have a stirring device. (4) Some provision should be made for collecting the volatile vanadium compounds. (5) An efficient hydrogen chloride generator is an important part of the apparatus.

After removal of the vanadium by this method, the next step is the conversion of the uranium compounds into urano-uranic oxide (U_3O_8). One of the commercial methods of conversion consists in boiling the sodium uranate with a concentrated solution of ammonium chloride. Ammonium uranate is produced, but the conversion is not complete. Upon ignition, the ammonium uranate is decomposed, yielding uranium oxide. A mixture of uranium oxide and sodium uranate is thus obtained, from which the sodium uranate is removed by digesting with dilute sulphuric or hydrochloric acids. From 10 to 15 per cent of the uranium oxide, however, is also dissolved. Hence the recovery of uranium oxide is only partial.

In this connection some experiments were conducted on the conversion of sodium uranate into uranium oxide. In one series of experiments both dry and moist samples of pure sodium uranate were dissolved in a slight excess of concentrated sulphuric acid and then preciptated with an excess of ammonia. After ignition and extraction with dilute sulphuric acid 59 per cent of the uranium was recovered as pure oxide U_3O_8 and in the case of moist uranate, the recovery was 64 per cent. This process resolves itself into virtually treating the sodium uranate with ammonium sulphate. Another experiment consisted in boiling both dry and moist sodium uranate in concentrated solutions of ammonium chloride, filtering off the precipitates, igniting and treating the residue with dilute sulphuric acid.

In the experiments with the dry sodium uranate by this method, the recovery of uranium oxide was found to be 64 per cent, and in the case of moist sodium uranate, a recovery of 61 per cent was obtained. These experiments show that in one of the commercial methods of converting sodium uranate into oxide approximately 64 per cent of the uranium is recovered, and assuming that 15 per cent of the uranium oxide is dissolved by the acid used in extracting the sodium uranate, it is seen that 75 per cent of the sodium uranate is converted into ammonium uranate.

Treatment of Carnotite Ore with Hydrogen Chloride to Remove Vanadium

Since hydrogen chloride removes vanadium completely from crude sodium uranate, the idea suggested itself of trying the effect of the gas on the vanadium in the original ore, with a view of developing a quantitative method of determining the vanadium in carnotite ores.

A few charges were run at room temperature, but it was found that the removal of the vanadium was far from complete, thus confirming the conclusions of Hillebrand.[2] Experiments were next tried by heating the ore while the hydrogen chloride was passing over it. It was found that the yield of chlorides and oxychlorides was much greater, but even in this case, only about one-half of the vanadium was removed. The ore caked rather badly, due to the water formed in the first part of the operation, so it was necessary to remove it from the tube, pulverize it and treat it again with hydrogen chloride.

The experiments which yielded the best results were conducted as follows: the sample of the ore was placed between plugs of glass wool in a hard-glass tube and hydrogen chloride passed over it until no more brown fumes were given off. The hydrogen chloride generator was disconnected then and the ore heated to drive off all moisture present. After this it was removed from the tube, thoroughly ground and mixed with powdered glass in order to let the gas have an opportunity to come in direct contact with each particle of the ore. The sample was replaced in the tube and hydrogen chloride (gas) passed over it for about two and one-half hours, at as high a temperature as could be obtained in the combustion furnace. The residue that remained in the tube was brown and sandy-like in appearance and gave only a very slight test for vanadium. Other experiments conducted under nearly the same conditions failed to give as satisfactory results as the one above did, showing that it is rather difficult to remove the vanadium by this method.

The treatment of the ore by hydrogen chloride for the removal of vanadium has a very direct bearing on the problem at hand. If the vanadium could be removed from the ore by this treatment at the beginning of the process a uranate would be obtained free from vanadium. Moreover, the removal of the vanadium before treatment of the ore for the extraction of radium would very materially simplify the entire process.

Treatment of the Crude Uranate with Chlorine Gas

In connection with the experiments of hydrogen chloride for the removal of vanadium, chlorine gas suggested itself as a possibility. Experiments were tried substituting chlorine gas for hydrogen chloride, but the results were negative. Chlorine gas does not react with the vanadium present in the crude sodium uranate, and only a minute trace of the tetrachloride and oxychloride of vanadium was observed.

Separation of Vanadium by Means of Ammonium Chloride

Inasmuch as hydrogen chloride readily removes vanadium from crude sodium uranate as a volatile product, it seemed worth while to conduct some experiments with mixtures of ammonium chloride and sodium uranate at somewhat elevated temperatures. Under these conditions the partial dissociation of ammonium chloride into ammonia and hydrogen chloride would furnish the constituent to react with the vanadium, and at the same time the ammonia and excess of ammonium chloride furnished favorable conditions for the conversion of some of the uranate into the oxide.

A number of preliminary experiments were conducted to ascertain: (1) quantity of water to have present in the mixture of uranate and ammonium chloride, (2) the temperature at which the reaction should proceed, (3) the amount of ammonium chloride required, (4) the kind of a vessel best adapted for the experiments, (5) the effect of other ammonium salts when added as partial substitutes for ammonium chloride.

Experiments were first conducted by mixing the crude uranate and dry ammonium chloride and subjecting this

[2]Bulletin No. 70, U. S. Bureau of Mines, 82 (1913).

mixture to heat in a test tube. The ammonium chloride was readily volatilized but apparently no vanadium was volatilized. A part of the uranate, however, was converted into the oxide. Experiments were then tried with the mixtures of ammonium chloride and crude uranate made up to a thick paste by the addition of water. When the paste is subjected to heat the water is gradually expelled, and then as the temperature is increased copious fumes of ammonium chloride are given off, and just as the volume of chloride vapor diminishes the characteristic brown vapors of the volatile vanadium compounds make their appearance and continue to be evolved copiously until the molten mass in the crucible becomes solid. Upon testing the residue obtained in this treatment, it was found to contain only a trace of vanadium.

The experiment was here tried of pouring the molten liquid from the crucible into a beaker containing water, to ascertain whether mere fusion with ammonium chloride would render the vanadium soluble. The melt dissolves almost completely in the water. The solution has a decided green color and gave test for both uranium and vanadium. Evidently this procedure must be included in the list of negative experiments.

Experiments were next conducted to determine the minimum quantity of ammonium chloride required for the nearly complete separation of vanadium by the above procedure. Mixtures of uranate and ammonium chloride in the following proportions were heated: (1) ten parts by weight of ammonium chloride to ten parts by weight of crude uranate, (2) fifteen parts by weight of ammonium chloride to ten parts by weight of crude uranate, (3) twenty parts by weight of ammonium chloride to ten parts by weight of crude uranate. The vanadium was satisfactorily separated only in the case where twice as much ammonium chloride was used as crude sodium uranate.

The experiments were first conducted in iron crucibles, but it was noted that a crucible would not stand more than two charges before it showed holes. Nickel crucibles were attacked fully as badly. Iron test tubes made out of 1¾-in. steam pipe capped at one end only withstood two charges. Alundum and porcelain crucibles, however, served the purpose very well, and in the experiments that follow porcelain crucibles were used.

In a few experiments, ammonium nitrate was substituted for a part of the ammonium chloride usually used. This modification did not yield very satisfactory results. While it was possible to remove the vanadium by using a mixture of fifteen grams of ammonium chloride, five grams of ammonium nitrate and ten grams of sodium uranate, the percentage conversion of uranium into oxide in the residue was thereby reduced, as shown in the tabulated data that follow.

As a result of these preliminary experiments the following procedure was adopted: A known weight of sodium uranate, ten grams in most cases, was taken and twice that weight of ammonium chloride added. The mixture is placed in a mortar and pulverized, which at the same time mixes it. The pulverized mixture is then placed in a porcelain crucible and water added until a stiff paste is obtained. Heat is then applied slowly at first with constant stirring to drive off the excess of moisture. Strong fumes of ammonia are noted at this point. As the mass becomes dry it has a tendency to cake, but with vigorous stirring this can be prevented to a large extent. After the water has been expelled the mass begins to fuse and continues to do so until there is a green molten liquid in the crucible. This liquid boils freely with the expulsion of ammonium chloride fumes until the excess of ammonium chloride is driven off and then the brown fumes of the chloride and oxychloride of vanadium begin to make their appearance. They were given off very copiously toward the end of process and especially as the molten mass became solid. Sometimes these brown fumes make their appearance before solidfication begins, but usually not. As soon as the volatile products cease to be given off, the crucibles were removed from the flame because further heating of the residue greatly increases the percentage of sodium uranate in the residue.

The black residue from this treatment gives a slight test for vanadium. The percentage of vanadium may be estimated colorimetrically by comparison with standard solutions of vanadium. In this way it was found that the vanadium content of the uranate is reduced from 8.55 per cent vanadium oxide to less than 0.5 per cent.

To ascertain to what extent the uranium is converted into the oxide by treatment with ammonium chloride, several quantitative experiments were conducted. The weight of the residue after treatment with ammonium chloride was first determined by introducing the ammonium chloride and a weighed quantity of uranate mixture in a tared crucible and making a run. From the weight of the residue thus obtained, the ratio of residue to uranate was found to be 0.958.

To ascertain the percentage conversion and the percentage of recovery of uranium, analyses were made of the residues obtained in several experiments. The residue from the crude uranate obtained by treatment with ammonium chloride must consist of a mixture of uranium oxide, sodium uranate, sodium chloride, iron oxide and impurities from the ammonium chloride. This residue was shaken with dilute (1:40) sulphuric acid in excess, which dissolves out the sodium salts of uranium and chlorides, and a small percentage of uranium oxide, leaving a black residue consisting of uranium oxide with a small amount of iron oxide. For commercial use the iron present is usually not objectionable.

Table II shows the results of a quantitative determination made on a portion of the residue, obtained by treating a given weight of crude uranate with twice its weight of ammonium chloride to determine the percentage of recovery of uranium oxide and the percentage conversion.

Column 2 represents the weight of the residue taken from a charge and used for analysis. Column 3 represents the equivalent weight of crude uranate calculated from the ratio 0.958. Considering the crude uranate contains 64.6 (Table I) per cent uranium oxide, the values in column 4 are deduced and represent the total amount of uranium oxide present in the residue. Column 5-a represents the weight of uranium oxide found after treating the residue with dilute (1:40) sulphuric acid. However, it was found, as will be shown later, that this residue is only 95 per cent pure, and thus b represents the weight of uranium oxide actually recovered on this basis. Column 6 represents the amount of uranium oxide present in the residue before treatment with dilute sulphuric acid. Assuming that 15 per cent of the uranium dissolved in the dilute acid used to remove the remaining sodium uranate. Column 7 repre-

Table II—Recovery of Uranium Oxide

No. of Sample	Weight of Residue	Crude Uranate Equivalent	Total U as Oxide	Weight of Pure U_3O_8	Total Conversion	Per Cent Recovery	Per Cent Conversion
1	0.5918	0.6184	0.3995	a—0.2840 b—0.2692	0.3174	67.4	79.2
2	0.6082	0.6355	0.4105	a—0.2930 b—0.2778	0.3268	67.6	79.6

sents the percentage recovery, figured from the weight or uranium oxide given in b, column 5. Column 8 represents the percentage conversion based upon the values of column 6.

The purity of the uranium oxide removed was determined by dissolving two samples of the uranium oxide residue after treatment with dilute sulphuric acid in nitric acid. The insoluble material was filtered off, and the iron precipitated with ammonium carbonate in excess. The iron was filtered off and the ammonium carbonate solution boiled to expel the excess of ammonia and precipitate the uranium compound, which was then converted to uranium oxide by ignition. In this way it was found that the residue contained 94.8 per cent U_3O_8.

To ascertain the effect of temperature on the percentage recovery of uranium, an experiment was run in which the final temperature was raised much higher than in the preceding case. Only 58 per cent of uranium oxide was recovered and the conversion was reduced from 79.5 to 68.3 per cent.

To confirm this conclusion a third series of experiments was conducted in which the volatile products were vaporized at still lower temperatures than obtained in the first experiment. The recovery of uranium oxide was raised to 78.5 per cent and the conversion attained the value 92.4 per cent. Hence the lowest temperature possible to remove the vanadium gives the most efficient recovery of uranium oxide.

A quantitative determination was made on the residue obtained by treating ten grams of crude uranate with a mixture of fifteen grams of ammonium chloride and five grams of ammonium nitrate. The vanadium here was removed to less than half a per cent, but the results of the experiment clearly showed that the percentage recovery, 48.8 per cent of uranium oxide, is very materially decreased. While the addition of the ammonium nitrate reduces somewhat the temperature at which the mixture fuses, it is evident its presence materially reduces the yield of uranium oxide.

Separation of Vanadium by Means of Mineral Acids

A—NITRIC ACID

Hillebrand refers[3] to the work of Friedel and Cummenge[4] on the separation of vanadic acid from carnotite ore by rendering the oxide insoluble in water by evaporating the nitric acid solution to dryness. This method of separating the vanadium from crude sodium uranate was tried, but it did not work very successfully, as a large part of the uranium was rendered insoluble as well as the vanadium, and then, too, the vanadium was not completely removed from the uranate.

Some modifications of this method were then tried, which finally furnished a quantitative method of separating vanadium from the crude uranate. In connection with some qualitative experiments upon the action of nitric acid on sodium uranate, we found that when the vanadium bearing uranate was dissolved in dilute nitric acid and the solution then boiled that the vanadium was precipitated quantitatively.

Quantitative experiments were then conducted to determine the following factors: (1) the proper amount of nitric acid necessary to dissolve the crude uranate, (2) the recovery of uranium from the solution of uranium oxide and its purity, (3) the amount of uranium in the precipitate containing the vanadium.

1. *The Quantity of Nitric Acid.*—By taking weighed samples of uranate and dissolving them in nitric acid of different concentrations and then boiling this solution to precipitate the vanadium, it was found that acids ranging in concentration from N/20 to N/50 completely removed the vanadium. Boiling of concentrated nitric acid solutions of the uranate results in only partial precipitation of the vanadium. The quality of pure nitric acid necessary for a given weight of crude uranate is given in the table below. It might be stated here that it is only necessary to boil the nitric acid solution for two or three minutes to precipitate the vanadium. It should also be stated that the quantity of nitric acid required is practically that shown by the equation,

$$Na_2U_2O_7 + 6HNO_3 = 2\ Na\ NO_3 + 2\ UO_2(NO_3)_2 + 3H_2O$$

2. *Recovery of Uranium and Its Purity.*—The recovery of the uranium from the filtrate of the nitric acid solution is best accomplished by precipitating it with ammonium hydroxide and then igniting the precipitate of ammonium uranate. Some sodium uranate is probably precipitated out with the ammonium uranate, which, upon ignition gives a mixture of uranium oxide and sodium uranate along with a small quantity of iron. The sodium uranate can be easily removed by agitating the ignited residue with cold dilute (1:40) sulphuric acid. The oxide of uranium thus obtained is free from vanadium, and sodium uranate.

It was found on analysis of the uranium oxide obtained after treatment with dilute sulphuric acid that it was about 95 per cent pure. The impurity here is undoubtedly due largely to iron present, which is not objectionable for most commercial purposes.

3. *Uranium Present in Precipitate Containing Vanadium.*—The amount of uranium present in the vanadium precipitate was determined in a manner similar to that used in determining the uranium in carnotite ore except that dilute hydrochloric was used to dissolve the precipitate. Concentrated hydrochloric acid and aqua regia do not dissolve it completely. The analytical data obtained are summarized in Table III. The uranium content given in column 3 was deduced from the analysis given in Table I. Column 4 gives the weight of insoluble residue containing the separated vanadium and the next column shows the weight of uranium oxide present in the residue. The figures in the last column show that about 14 per cent of the uranium is carried down in the residue. The further treatment of this residue for the separation of uranium and vanadium is in progress.

TABLE III—ANALYSIS OF VANADIUM PRECIPITATE

No. of Sample	Weight of Crude Uranate	Total U as Oxide	Weight of V_2O_5 Residue	Weight of U_3O_8 in Residue	Per Cent U_3O_8 in Residue	Per Cent U_3O_8 of Total U_3O_8
1	2.8270	1.826	0.5728	0.2506	43.7	13.2
2	2.9383	1.898	0.5590	0.2733	48.9	14.4

Next a series of determinations was run to find the weight of pure nitric acid required to react with a given weight of crude uranate, the percentage recovery of uranium oxide and the weight of the vanadium residue. Four determinations were made and the results obtained are given in Table IV. The numbers in the fifth column represent the weights of crude uranium oxide recovered and the next column gives the weights of uranium oxide after treatment with dilute sulphuric acid to dissolve out sodium uranate.

It will be noted from this table that the recovery of uranium dropped considerable in the cases of samples 2 and 4. This result is readily accounted for when the sulphuric acid filtrate (column 6) was taken and analyzed for uranium oxide. In the case of sample 2, 22.5

[3]Bulletin 262, U. S. Geological Survey—Contributions to Mineralogy, 22.
[4]Compt. rend. 125, 532 (1899).

TABLE IV

Number of Charge	Weight of Crude Uranate	Total U as Oxide	Weight of Pure HNO_3 per G. of Uranate	Weight of U_3O_8 by Pp't	Weight of U_3O_8 Obtained After Treatment with Dilute H_2SO_4	Weight of V_2O_5 Residue
1	3.6197	2.3383	.....	2.1320	1.8530	.....
2	2.8270	1.8262	0.6048	1.5986	1.0354	0.5728
3	2.9383	1.8980	0.5738	1.6173	1.2997	0.5590
4	10.5883	6.8440	0.5719	5.9026	4.0116	2.2349

					Per Cent V_2O_5 Residue	Per Cent U_3O_8 Recovered
1					....	79.2
2					20.2	56.6
3					19.0	68.4
4					21.1	58.6

per cent of the total uranium oxide was found in the filtrate and 25 per cent for sample 4. It will also be noted that the weight of vanadium residue bears an almost uniform ratio to the weight of the crude uranate treated. This method of treatment still leaves approximately 13.5 per cent of the total uranium in combination with the vanadium, but the vanadium is completely removed and 65 per cent of the uranium is recovered as pure oxide.

B—TREATMENT WITH HYDROCHLORIC AND SULPHURIC ACIDS

After obtaining a good separation of vanadium from crude uranate with dilute nitric acid, the investigation was extended to dilute hydrochloric and sulphuric acids. Experiments were conducted very similar to those used in the case of nitric acid, and it was found that here, again, the vanadium could be removed quantitatively with hydrochloric acid, but only in some experiments was the treatment with sulphuric acid successful.

Two samples of the crude uranate of approximately five grams were dissolved in the least possible amount of hydrochloric acid, which had been standardized. The solutions were then brought to a boil and boiled for a few minutes to precipitate the vanadium which settled out very readily. The vanadium residue was filtered off and ammonia added to precipitate the uranium. The precipitate of ammonium uranate was filtered off, ignited to uranium oxide and the residue treated with dilute (1:40) sulphuric acid to extract the sodium uranate present in the ignited residue.

Table V gives a summary of the results obtained in these experiments.

TABLE V—SEPARATION OF VANADIUM WITH HYDROCHLORIC ACID

Number of Charges	Weight of Crude Uranate	Total U as Oxide	Weight of Pure HCl per g. of Uranate	Weight of U_3O_8 by Pp't	Weight of U_3O_8 from H_2SO_4 Residue	Weight of U_3O_8 in H_2SO_4 Filtrate	Weight of V_2O_5 Residue
1	5.7852	3.7383	0.3472	3.3586	2.3864	0.8884	1.0160
2	5.4948	3.5496	0.3712	3.2353	2.4673	0.7078	1.2520
		Per Cent		Per Cent	Per Cent	Per Cent	Per Cent
1		64.63		89.8	63.3	23.8	18.4
2		64.63		91.1	69.5	19.9	22.7

Comparing the percentage of vanadium oxide residues here obtained with those obtained by the nitric acid treatment, it appears that the percentage weight of the vanadium bearing residues averages 20.5 per cent, which is in good agreement with the percentage obtained by the nitric acid treatment. Hence we assume that the vanadium oxide residue is the same in both cases.

It will also be seen, from the table, that the percentage of pure uranium oxide in sample 2 is greater than in sample 1, but, on the other hand, the amount of uranium oxide recovered from the sulphuric acid treatment is less, which accounts for the difference. It thus appears that the two acids serve equally well.

A number of experiments have been conducted with sulphuric acid of different concentrations ranging from fifth normal to fortieth normal following the same procedure as with dilute nitric and hydrochloric acids. Thus far we have met with but partial success. In a few instances out of a number of trials the characteristic brown vanadium bearing precipitate was obtained, and the separation of vanadium was practically complete. The quantity of uranium in the precipitate was found to be practically the same as in the precipitate obtained by treatment with nitric acid.

More frequently, however, when the dilute solution of sodium uranate was boiled a canary yellow precipitate appeared. The quantitative experiments conducted made it evident that a part of the vanadium remains in solution whenever the canary yellow precipitate comes down. If we can succeed in duplicating some of the quantitative results obtained in a few cases the treatment with sulphuric acid would be ideal as the cost of sulphuric acid is considerably less.

Summary of Results

The experiments here described have resulted in at least four methods that have proved successful for removal of vanadium from the crude uranate, which is obtained on a large scale in the plant of the National Radium Institute.

1. By the action of hydrogen chloride gas it has been shown that in time the vanadium is completely removed, leaving a residue consisting of sodium uranate, sodium chloride, uranyl chloride, and the other impurities in the crude uranate.

The uranium may be removed from this residue by either of two methods: (1) By boiling with excess of ammonium chloride, which converts the uranium into ammonium uranate, which may be ignited to uranium oxide; (2) by dissolving the residue in a dilute mineral acid and precipitating the uranium with ammonia, as ammonium uranate, which may be converted into uranium oxide by ignition.

The percentage recovery of uranium oxide, free from vanadium, but containing a small percentage of iron, which is not objectionable for its sale, varies from 59 to 64 per cent.

2. By mixing the vanadium bearing uranate with twice its weight of ammonium chloride and enough water to make a thick paste, the vanadium may be nearly completely volatilized by heating in a porcelain crucible. When subjected to proper conditions of temperature, it is possible to reduce the quantity of vanadium present in the crude uranate from 8.5 per cent to less than ½ of 1 per cent. The ammonium chloride serves the double purpose of converting a part of the uranium present to ammonium uranate, which, upon ignition, yields uranium oxide as well as removing the vanadium.

The percentage of uranium recovered may be regulated very materially by the heat treatment to which the mixture is subjected. Where the residue, after volatilization of the ammonium chloride, is heated to a high temperature, the percentage recovery of the uranium oxide (U_3O_8) is reduced due to the fact that some of the uranium oxide formed in the operation is converted back to sodium uranate.

The highest percentage recovery was obtained where the temperature was not raised any higher than necessary to volatilize the ammonium chloride and vanadium compounds.

In this connection it was found that a mixture of fifteen grams of ammonium chloride and five grams of ammonium nitrate to every ten grams of crude uranate served the purpose of removing the vanadium from the crude uranate just as well as the ammonium chloride alone. However, the percentage recovery of uranium oxide is reduced very materially, as it is only possible to get a recovery of about 58 per cent under the most favorable conditions of temperature. This reduction in the percentage recovery more than counterbalances any advantage that the use of ammonium nitrate might have as a partial substitute for ammonium chloride.

3. The vanadium can be completely precipitated by dissolving the crude uranate in the least possible amount of dilute nitric, hydrochloric and sometimes sulphuric acid, and then boiling the solution. Approximately 13 per cent of the uranium is precipitated out with the vanadium, but the amount is small compared to that which remains in solution. The uranium may be recovered by adding ammonia to the solution containing the uranium, free from vanadium, to precipitate it as ammonium uranate, which may be converted into uranium oxide by ignition.

The percentage recovery of uranium oxide in the case of nitric and hydrochloric acid treatments is found to vary from 58 to 79 per cent. Some uranium is lost in the dilute sulphuric acid treatment of the residue for the removal of the sodium uranate, but this uranium may be recovered from the acid solution by precipitation with ammonia.

Only by one of the above methods is the removal of the vanadium and the conversion of the uranium oxide accomplished in one step. That is the case of the ammonium chloride treatment. In all of the other processes the vanadium is removed by one step and the uranium recovered by another operation. However, it will be noted that the percentage recovery of uranium oxide in all the methods was practically the same.

4. Probably hydrogen chloride (gas) may also be utilized to remove the vanadium from carnotite ores quantitatively, by proper regulation of temperature.

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The American Smelting & Refining Company's tin-smelting plant in the course of construction at Perth Amboy, N. J., will shortly be finished. The company will receive concentrates from Bolivia by way of the Panama Canal, smelt these in reverberatory furnaces with subsequent electrolytic refining. Some shipments of ore have been held up at the canal, which have delayed operations. Very pure tin analyzing 99.95 per cent has been produced in the experimental plant and it is the intention to make a tin to compete with Straits tin.

Geological Survey Cannot Make Assays.—The United States Geological Survey calls the attention of mining men and others to the fact that it is not equipped for making assays or analyses of rocks, its laboratory being in fact overloaded with official work. Many specimens and samples are received by the Survey, accompanied by requests for such treatment, with which it is impossible to comply. The force of chemists employed is small, and their time is fully occupied by their regular official duties. The Geological Survey has no facilities for making gold and silver assays. The most the Survey geologists can do is to give an off-hand opinion based on a simple examination of the specimens.

The Rubber Industry

BY ANDREW H. KING

It has been said that too frequent association dulls the appreciation and relegates important discoveries and advances to the rank of the commonplace. This statement holds very true about rubber. The most of us have grown up with rubber articles about us all our lives. In fact, our association with this remarkable substance dates from the nursing bottle. Consequently, it is not at all out of the way for us to accept rubber as a matter of course.

No material has yet been found which combines such a degree of strength, elasticity, and wearing power. These properties make rubber especially valuable in manufacturing the various grades of hose—air, water, steam, and oil; belts; valves and other molded articles, and tires of all kinds. A complete list of all the various things made of rubber would be almost infinite.

The first mention of this remarkable substance from a European was by Christopher Columbus, who on his second voyage of discovery (1493-1496) saw the natives of Haiti playing with balls, which he afterward learned were prepared from the gum of a tree. This gum they called cahuchu, from which the word caoutchouc is derived. Highly interesting must this gum have been to the Spaniards. By it they were able, in a crude manner, to waterproof their garments, and thus protect themselves from the heavy tropical rains. They also made the acquaintance of the related substance, balata. They found shields made of balata impregnated cloth. The same material was used in the preparation of suits which resisted poisoned arrows. In fact, such suits are now to be seen in the Spanish museum at Madrid. This was indeed a fine forerunner of the modern balata belt.

The English word rubber was given it by Priestly in 1770, who recommended its use for removing pencil marks. This would likely have remained its principal use had not the discovery of vulcanization been made. Raw rubber is not suitable for proofing garments, due to the narrow range of temperature within which it is elastic but not sticky. Below 10 deg. C. it begins to become hard, and at 0 deg. C. it is brittle. However, it regains its former properties on warming. Above 25 deg. C. it has a great tendency to become sticky, and if heated above 75 deg. C. it becomes permanently tacky.

But in 1839 Nelson Goodyear, an American, made his important discovery of vulcanization. The beginning of the rubber industry dates from this invention. The bicycle craze of some years ago gave it added impetus, and as the cycle went out automobiles came in. It is now an accepted fact that these machines cannot be comfortable without pneumatic tires. It has been estimated that at least 85 per cent of all rubber used in manufacturing goes to tires, and only 15 per cent to mechanical goods. The United States alone uses about 100,000 tons of crude rubber per year. So we see that rubber is an established industry.

The Varieties of Crude Rubber

Schidrowitz divides the important rubber-producing trees into four orders, with the following subdivisions:

I. Euphorbiceae—
  1. Hevea.
  2. Manihot.
  3. Sapium.
  4. Micandras.

II. Apocynaceae—
  1. Funtumia.
  2. Landolphia.
  3. Clitandra.
  4. Hancoria.
  5. Dyera.

III. Urticaceae—
  1. Ficus.
  2. Castilloa.

IV. Compositae—
  1. Guayule.

Hevea, brasiliensis.—This is by far the most important rubber of the whole family. It supplies over 60

FIG. 1—HALF HERRING-BONE SYSTEM

FIG. 2—HERRING-BONE SYSTEM

per cent of the total output. It was originally native to South America, where great forests of these trees abound. They are often of great height and girth. From the experience planters have had with Hevea in the East, the age of these trees must be well over 500 years. For these reasons alone they should produce the best grade of rubber. In addition, the method of curing or coagulating the latex, while very primitive, gives a product far superior to that secured from cultivated trees. The Amazon valley is the principal rubber district of South America. Commercial Brazilian rubbers are usually classified as Up-river or Hilands, and Islands. The former is shipped from Manaos, which is about a thousand miles up the Amazon. The latter comes from the City of Para, on the coast. Hilands is sometimes called hard cure and Islands soft cure. The commercial varieties include also Fine Para, Coarse Para, Caucho, Matto Grosso, Peruvian, and Bolivian.

The Hevea brasiliensis has been transplanted to the East Indies. Climatic and soil conditions proved suitable, and the wild rubber tree has yielded to cultivation. Now great quantities of rubber are produced on such islands as Ceylon, Borneo, Sumatra, and on the Malay Peninsula. Many varieties of plantation sheets are on the market.

Manihot glaziowii.—This variety is native to Brazil, but has been planted with success in parts of Africa, where Hevea brasiliensis proved a failure. The most important commercial variety of the species is Ceara or Manicoba.

Funtumia elasticia.—This species is native to tropical Africa, and supplies the rubbers know as Gold Coast Lumps, and some of the Congo and Cameroon varieties.

Landolphia, owariensis, etc.—These species include various vines and creepers which occur in enormous quantities in Congo and parts of the East and West Coast, and in Central Africa. The vines are ordinarily cut down and the latex secured by bleeding. It furnishes Upper Congo balls, Sierra Lionne Niggers, Red and Black Kassai, and Madagascars.

Hancoria speciosa.—This species is native to Brazil and central South America. It yields Mangabeira, Pahia, and Matto Grosso.

Dyera costulata.—This supplies the very poor rubbers known as Pontinak, Jelutong or Dead Borneo. They contain but about 15 per cent of rubber, the remainder being resin, dirt and water. This variety comes from parts of the Malay Peninsula, Sumatra, and Borneo. Considerable quantities of it are purified. The resin finds many uses, and is sold under the name Malaysian or Pontinak resin. Formerly the trees were felled and bled where they lay on the ground. This very wasteful method has almost entirely given place to scientific tapping. The rubber extracted from Ponti is very good.

Ficus elasticia.—This species is found principally in Asia. Burmah, Java and India are the important districts. It has been planted to some extent in the Dutch and German African colonies.

Castillai elasticia.—This variety is indigenous to Mexico, Central America and Peru. It produces Peruvian Caucho, Mexican Strips and Centrals.

Parthenium argentatum.—From this species is obtained Guayule. It is found principally in Mexico and southern Texas. The rubber occurs in the solid state among the woody fibers of the shrub. Guayule found considerable favor in this country until continued revolutions made the gathering of the shrub and the extraction of the rubber at least rather hazardous, if not to say impossible. While the production of Guayule destroys the shrub, it does not seriously cripple the industry, since it grows very fast and was formerly looked upon as farmers regard blackberry bushes.

For 1912 Maclaren made the following estimate, which will serve to show the relative importance of the various grades of rubber. In these figures allowance has been made for the resin content of low-grade rubbers.

Plantations	28,500 tons
Brazil (Amazon Basin)	48,830 tons
Brazil (Bahia)	4,000 tons
Central America	2,500 tons
Africa	20,000 tons
Guayule	3,500 tons
Jelutong	6,000 tons
Total production	113,330 tons

FIG. 3—SERINGUEIRO SMOKING RUBBER AT ALTO ACRE

Basing his figures on the increase in production of plantation rubbers, the decreasing supply of some wild rubbers, and upon the further demand, he estimates for 1916 as follows:

Brazil	15,500 tons
Central America	250 tons
Africa	1,000 tons
Jelutong	500 tons
Plantations	130,000 tons
Total production	147,250 tons

While the figures for 1915 are hardly yet available, it seems that his estimate will at least prove conservative. For 1920 he estimates as follows:

Brazil	2,000 tons
Various	500 tons
Plantations	160,700 tons
Total production	163,200 tons

According to his opinion the cheaper grades of rubber will be gradually eliminated, and there will remain only Brazilian and plantation rubbers. Another factor in driving out the low-grade rubbers is the improved shoddy processes, which furnish cheap and fairly good material. In fact, many rubber chemists prefer high-grade shoddy to some of the low-grade rubbers.

The following figures were taken from an extensive table by Spence, and serve to show the relative values of crude rubbers on the basis of their washing loss and resin content.

Trade Name	Washing Loss, Per Cent	Resin Content, Per Cent
Fine Para, Islands	17 -20	2 - 3
Fine Para, Hilands	14 -19	1.6 - 3
Plantation Crepe	0.2 - 1	1.3 - 3.6
Caucho	18 -23	3.5 - 5.5
Manicoba	30 -55	3 - 6.5
Gold Coast Lumps	30 -55	7 -10
Jelutong	64 -70	70 -85
Guayule	15 -40	20 -25

Tapping

Rubber is derived by a process of coagulation from the latex of certain trees, shrubs, or plants, which have been already discussed. The latex is quite distinct from the sap, and is contained in a separate cell system which lies between the outer bark and the cambium. Obviously a very thorough knowledge of the lactiferous system is necessary to yield the best results on tapping.

In the Amazon district the old V method is still in use. It consists of a series of V's, the first being about 6 to 7 ft. above the ground, and successive cuts beneath it down the trunk. The next row begins about 18 in. over, and just a little lower than the first one, and so on down the tree. The third row begins with a cut on a level with the first one. Earthenware cups are placed at the apex of each V to collect the latex.

In the plantation districts of the East the half herring-bone method is most preferred. This consists of a series of oblique cuts running into a central channel. See Fig. 1. Additional tapping is done by paring away a strip of bark about ¼ in. wide along each cut. When a sufficient area is exposed the tapping is transferred to the opposite quarter of the tree. When the allowable area of trunk is exposed on that side, the tree is left to rest for three or four years. Later the tapping is begun on the quarters adjacent to the former wounds. The full herring-bone method also finds favor, and consists of simply twice the area of the half herring-bone. See Fig. 2. The latex is collected in a small iron cup fixed at the base of the central channel. These cups are emptied at regular intervals, and the latex taken to the plantation factory for coagulation. There are other ways of tapping, but these are the most important.

The Nature of Rubber Latex

Latex is a white to cream-colored fluid, resembling milk in appearance. When this emulsion is examined under the ultra-microscope it is found to consist of very minute (0.5 to 4 mu.) oil-like colloidal particles floating in a clear serum. They are in rapid Brownian movement, and when the latex is electrolized they go to the positive pole, thus proving them to be negatively charged colloids.

The liquid is either slightly alkaline or amphoteric in reaction. It contains various enzymes, which if allowed time to act will produce acids, and thus precipitate the colloids. Besides India rubber, the latex contains vari-

ous resins, proteins, and acid salts, such as sulphates, malates, phosphates, and chlorides. Sugars and alkaloids may also be present. Seelingmann gives the following average analysis of Para latex:

	Per Cent
India rubber	32
Albumenoid extracts and mineral matter	12
Water	56

Coagulation

By coagulation we mean the formation of an irreversible colloidal gel. The colloids coalesce. The particles grow larger and larger, and begin to throw out minute hair-like arms and connect one another. This leads to a reticulated or network structure which is not lost when the sheet is ready for market. Schidrowitz coagulated a drop of latex between a slide and cover glass, and noticed that the reticulated appearance could be intensified and made coarse by working a little and by using slight pressure. Fig. 4 illustrates this phenomena. This gave him an explanation of the effect of working on freshly coagulated latex. Latex coagulated by chemical means appears first as a cheesy mass, which readily gains in strength and cohesiveness on working through the rolls.

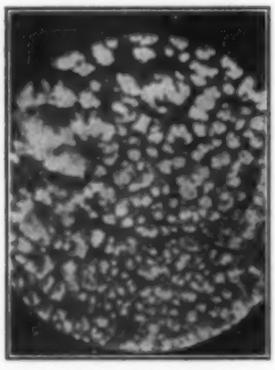


FIG. 4A — MICRO-PHOTOGRAPH OF A DROP OF FRESHLY COAGULATED FUNT. ELAS. LATEX, SHOWING RETICULAR STRUCTURE

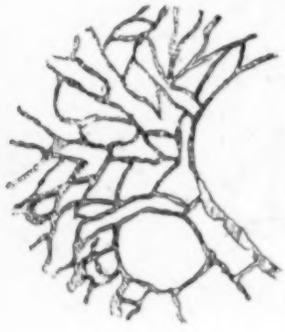


FIG. 4B—THE SAME AS "A" AFTER APPLYING PRESSURE. NOTE THE ALTERATION IN STRUCTURE. (FROM A DRAWING.)

Coagulation has been performed by many methods. In this paper I will only discuss those most in use and which give the best results. The market value of a rubber depends much on the care used during coagulation. For example, African rubbers are as a rule very dirty, highly resinous, and of a very foul odor. No care is used in gathering the latex; in fact, many of the savages smear it on their stomachs until they have a good-sized lump, and then peel it off. The heat of their bodies coagulates it.

Probably the oldest and undoubtedly the best method of coagulation is that used in the Amazon country. The "seringueiro" or rubber gatherer first makes a small fire, in which he burns the nuts of the urucuri palm. These nuts give a dense smoke, which is rich in the products of wood distillation, such as creosote, acetic acid, and so forth. Next he places a long pole or mandril over the fire, and pours on it a small quantity of latex. The pole is rotated until a thin film of dried rubber has formed, when he makes another application. The process is repeated until he has a loaf of about 100 lb. Rubber so prepared forms the biscuits of trade, which are easily recognized since the loaf is made up of concentric layers which may be separated. Fig. 3 shows the hut of the seringueiro. There is no rubber better than Hilands Fine Para. The method of coagulation is largely responsible for this, since it allows all the elements of the latex to be evenly distributed throughout the mass.

On the plantations coagulation is brought about by the addition of a small quantity of very dilute organic acid. Acetic is mostly used, though formic also finds favor. It has become the custom to add a small quantity of sodium sulphide to the latex before coagulating. This acts as a preservative, and prevents enzymeic action which would otherwise cause a black rubber. Most of the Eastern sodium sulphide rubber is sold under the name of Ceylon, and is a very nice, white, clear rubber, greatly in favor, if not absolutely essential, to the manufacturer of white articles. The requisite quantity of acid is added, and the latex allowed to stand a short time, during which coagulation occurs. The freshly prepared rubber is worked and washed on corrugated rolls. This removes any serum left in the pores, and also any foreign matter.

At this point it is well to mention that this excessive cleanliness is a detriment to plantation rubber, for it removes certain nitrogenous bodies which greatly influence the time of vulcanization, and thus make necessary the employment of some form of accelerator. Accelerators are used also with Para, but are not nearly as necessary.

After washing, the rubber is sheeted on smooth rolls. It is then dried, either by means of vacuum dryers or by hanging in a loft. When dry it may be smoked. Hardwood is mainly used for this purpose, and it supplies much the same vapors as are used along the Amazon. The object is to disinfect the rubber and thus prevent bacterial action. Rubber so treated is called smoked sheets, and forms a large portion of the plantation output.

Kaye in a recent article gives the following amounts of various acids required to coagulate normal Hevea latex containing 30 per cent of rubber:

Acid	$\frac{N}{10}$ cc Acid to 100 cc. Latex
Sulphuric	20 cc.
Hydrochloric	9 cc.
Nitric	31 cc.
Acetic	150 cc.
Oxalic	31 cc.
Tartaric	33 cc.
Citric	71 cc.
Formic	20 cc.

He mentions the fact that no matter what acid is used for coagulation, after the process is complete there is always approximately twice the acidity in the case of Hevea latex than one should expect from the quantity of acid used. The increase is thought to be due to phosphoric acid produced by the hydrolysis of some of the protein groups.

Physical Properties of Rubber

As was previously stated, the structure of rubber is reticular. It is this property which makes possible the addition of large amounts of other substances, as is regularly done in compounding. Films of rubber prepared from solutions do not display this structure until treated with sulphur chloride, after which it is quite pronounced.

The specific gravity of clear washed and dried rubber is usually taken as 0.925. On warming, the gravity gradually decreases.

Rubber is not soluble in water, but will absorb as much as 45 to 50 per cent of it. It is insoluble in acetone or alcohol, consequently these solvents are mainly used in extracting the resinous constituents. It forms a typical colloidal solution with benzole, gasoline, chloroform, toluene, ether, etc. Fig. 5 is an ultra-microscopic photograph of a solution of rubber with benzole.

Chemical Properties

This subject is very broad, and not well established. A full discussion is beyond the scope of this paper.

Alkalies.—Neither aqueous nor alcoholic solutions of the alkalies effect it; i.e., below 100 deg. C. Prolonged digestion with ammonia causes india rubber to pass into an emulsion closely resembling the original latex.

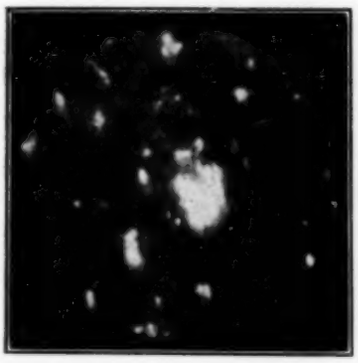


FIG. 5 — ULTRA-MICRO-PHOTOGRAPH OF A SOLUTION OF RUBBER IN BENZENE

Chlorine. — When chlorine is passed through a rubber solution a white powder having the formula $C_{10}H_{14}Cl_8$ is formed. This is evidently an addition and substitution product.

Bromine. — Bromine forms in the same manner two products, $C_{10}H_{16}Br_4$ and $C_{10}H_{16}Br_8$. The tetrabromide has been suggested as a means for the direct determination of the percentage of rubber in rubber analysis.

Acids.—*Hydrochloric* acid gas does not act on vulcanized rubber articles when absolutely dry. In the presence of moisture an additional product is formed.

Sulphuric acid when concentrated acts very vigorously, charring and oxidizing it.

Nitric acid, concentrated, attacks rubber energetically, forming first a yellow body which is subsequently decomposed into nitrogen, carbon dioxide, oxalic acid, and a substance of the character of a fat. On prolonged boiling this is converted into a mixture of camphoric and camphoronic acids.

Nitrous acid, when carefully prepared by running concentrated sulphuric acid into a 20 per cent solution of sodium nitrite and carefully dried by passing the gas through drying tubes containing phosphorous pentoxide, then passed into a solution of dry rubber in water-free benzole, a product having the formula $C_{10}H_{16}N_2O_4$ is formed. This is the rubber nitrosite, and is an amorphous, yellow compound which is insoluble in most of the ordinary solvents, but fairly soluble in dimethyl oxalate. This product has also been suggested for the direct determination of rubber.

Dry Distillation.—On dry distillation of purified Hevea, Weber obtained the following:

	Per Cent
Isoprene	6.2
Dipentene	46
Heveene	17
Polyterpenes	26.8
Carbon residue	1.9
Mineral residue	0.5
Loss (water and gases)	1.4

The Rubber Hydrocarbon

When commercial rubber is highly purified, it is found to consist almost entirely of a substance having the empirical formula $(C_5H_8)_n$ or $(C_{10}H_{16})_n$. The latter one is most accepted.

Commercial rubber also contains about 3½ per cent of a so-called "insoluble" constituent. Before Weber's study of the subject and his determination of the correct amount, it was thought to supply the strength, while the true rubber hydrocarbon gave the elasticity. Weber easily disproved this theory, and showed it to have the formula $C_{27}H_{42}O_{19}$. He suggested that it forms a link between rubber and the complex carbohydrates, as the celluloses, which he considers the raw material from which the plant produces all the terpenes, including rubber. He illustrates his theory by the following reaction:

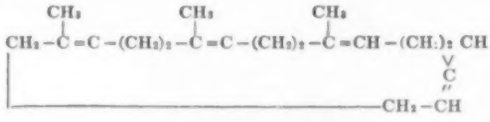
$$C_{27}H_{42}O_{19} = 3(C_{10}H_{16}) + 10H_2O$$

Harries, on the basis of his work on rubber ozonides, regards it as polymerized 1:5 dimethyl cyclo-octadien

$$\left[\begin{array}{l} CH_3-C(=CH)-CH_2-CH_2-CH \\ \qquad\qquad\qquad\qquad\quad\ \ \| \\ \qquad\qquad\qquad\qquad\quad\ \ C-CH_3 \end{array}\right]_n$$

Rubbers obtained from different sources are not always identical in structure. Harries found that while ozonides of purified Hevea gave on hydrolysis mainly laevulic aldehyde and laevulic acid as a secondary product, the reverse was true of the ozonides of gutta percha, which also has the formula $(C_{10}H_{16})_n$, and of certain African species. His explanation is that the ozonide complex occupies a different position relative to the plane of the eight-ring space, a cis position in one and a trans in the other.

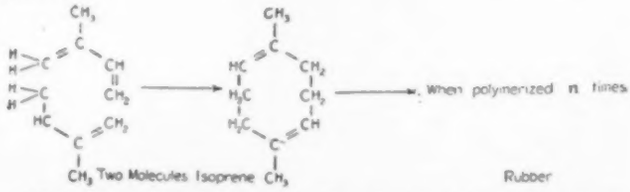
Pickels has suggested that this behavior be explained on the assumption that the rubber hydrocarbon does not consist of a number of polymerized $C_{10}H_{16}$ rings, but of a single ring containing a number of unsaturated C_5H_8 nuclei.



Synthetic Rubber

The production of synthetic rubber has long been the dream of the organic chemist. The conquest of alizarin and indigo gave him confidence to attempt this most difficult task. Synthetic rubber has already been produced, but not in quantities, nor at a price that can compete with the natural product. For synthetic rubber to be an assured success it must be cheap. It has been said that rubber can be laid down in this country from the East at 20 cents a pound should occasion demand it. The rubber grower has allied himself with the botanist, and each year he is strengthening his position. However, should cheap synthetic rubber come, we may well expect the rubber streets that have been predicted.

The beginning of the attempt to synthesize rubber dated from the researches of Harries into the nature of the rubber hydrocarbon. According to his theory, two molecules of isoprene polymerize to 1:5 dimethyl octadiene and this is further polymerized *n* times to rubber.



Natural rubbers seem to be derived from isoprene, but for simplicity and logic the synthetic chemist begins with butadiene, also called divinyl or vinylethylene $CH_2=CH-CH=CH_2$ since isoprene is but B methyl butadiene $CH_2=C(CH_3)-CH=CH_2$. One fact seems already proved. Whenever the grouping C:C·C:C can be obtained in a compound that compound is capable of being polymerized to rubber. By making substitutions in the butadiene hydrocarbon, it should be possible to secure a whole series of rubbers.

Two molecules of butadiene polymerize as indicated:

$$\left.\begin{array}{l} CH_2=CH-CH=CH_2 \\ CH_2=CH-CH=CH_2 \end{array}\right\} \rightarrow \left(\begin{array}{l} CH_2-CH=CH-CH_2 \\ \qquad\qquad\qquad\quad | \\ CH_2-CH=CH-CH_2 \end{array}\right)_n$$

In accordance with the above, dipropylene rubber should be formed:

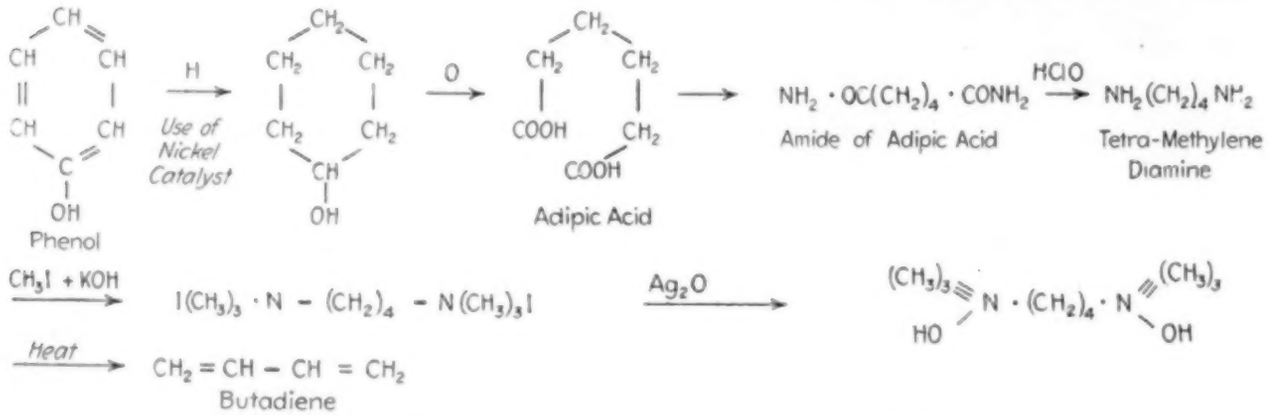
$$\begin{matrix} CH_3=C(CH_3)-C(CH_3)=CH_2 \\ CH_3=C(CH_3)-C(CH_3)=CH_2 \end{matrix} \longrightarrow \left[\begin{matrix} CH_2-C(CH_3)=C(CH_3)-CH_2 \\ | \\ CH_2-C(CH_3)=C(CH_3)-CH_2 \end{matrix}\right]_n$$

Thus we see that the first part of the problem is to make butadiene or its homologues cheaply. The second part is to efficiently polymerize this substance or substances to rubber. The solution of the second part is already nearly complete, but the first still seems difficult.

The following methods have been proposed for the production of butadiene and its homologues:

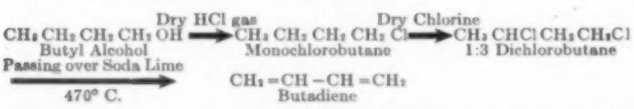
From Phenol

Hofman has used the method of synthesis outlined below. For butadiene he begins with phenol, and for isoprene he starts with paracresol. The reactions are identical, so I will indicate only those for butadiene.



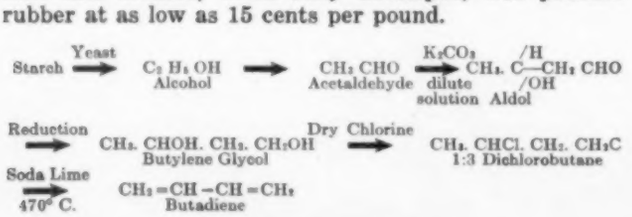
By Fermentation From Starch

Fusel Oil Route.—Fernbach has developed a form of bacteria which is capable of producing about 42 per cent of fusel oil from starch. By varying the conditions of fermentation he claims he can produce fairly large quantities of acetone as a by-product. The butyl alcohol of fusel oil is separated and purified. The steps in the process are indicated below.



By using iso-amyl alcohol, which forms a large percentage of the fusel oil obtained from cereals or potatoes, in place of butyl alcohol in the above indicated process isoprene can be obtained.

Acetaldehyde Route.—Perkin in his synthesis also begins with starch, which he ferments to ethyl alcohol by means of yeast. This is converted to acetaldehyde. From this he obtains aldol, which is converted to butadiene by the steps which are indicated below. He claims that this method, when fully developed, will produce rubber at as low as 15 cents per pound.



From Turpentine

The isoprene lamp has long been known. Harries and Gottlob prepared isoprene by its use quite early in their researches. It consists of nothing more than a flask, in which is suspended just below the neck a platinum coil, which is electrically heated to incandescense. The turpentine or other liquid is boiled in the flask. The vapors are decomposed by the coil and the isoprene is separated by fractionation. Herty and Graham obtained 5.5 per cent isoprene from spirits of turpentine in this manner. By trying the different fractions of commercial spirits of turpentine, they concluded that this is mostly due to pinene and not to dipentene, as was claimed by Harries and Gottlob.

By the use of the isoprene lamp, Harries and Gottlob also obtained as high as 50 per cent isoprene from commercial limonene. Since the volatile oil of pinus serotina contains considerable limonene, Herty and Graham used it in place of turpentine in the isoprene lamp. They obtained a yield of 12 per cent isoprene.

Turpentine as a raw material is too expensive for any of these processes to be a decided success.

Pyrogenic Reactions

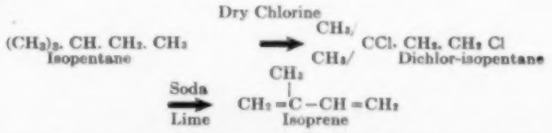
Some very excellent work has been done by Kyriakides on this side of the water. His process is essentially one of dehydration. For example, one method for the preparation of butadiene is as follows: beta-butylene glycol is slowly dropped into a tube containing pieces of ignited kaolin heated to 380-400 deg. C. Butadiene is obtained. The dehydration, he thinks, takes place thus:

$$CH_3-CHOH-CH_2-CH_2OH \longrightarrow CH_3-CH=CH-CH_2OH+H_2O$$
$$CH_3-CH=CH-CH_2OH \longrightarrow CH_2=CH-CH=CH_2+H_2O$$

The reader is referred to his original work, as published in the *Journal* of the American Chemical Society, Vol. 36, pages 530, 657, 663, 980, 987.

From Petroleum

Petroleum has been suggested as a source of butadiene, and its homologues, since by careful fractionation isopentane, $(CH_3)_2CH\cdot CH_2CH_3$, can be obtained. This may be converted to isoprene by the following steps:



The above are the principal published methods for the production of butadiene and its homologues.

As before stated, the second part of the problem consists in efficiently polymerizing these substances to rubber.

Tilden was the first to discover that isoprene could be converted to rubber. In 1882 he examined a tube of isoprene which had been sealed and set aside twenty

years previously. He was surprised to find its contents no longer liquid. This substance he took to be rubber. It has been said that he did not sufficiently prove that his product was rubber, however there seems to be no doubt that he was right.

If isoprene is spontaneously converted to rubber as the above would indicate, then catalysers which would hasten the reaction should exist. This fact led to the trial of many substances, such as acetic anhydride, hydrochloric acid, etc. Different observers at times obtained polymerization products, but on the whole their experiments could not be satisfactorily duplicated by others.

However, in 1912 Matthews tried the action of sodium wire on isoprene, and was successful. He used about 5 per cent of sodium and kept his sealed tubes slightly warm. This discovery proved to be the key to synthetic rubber, for it provided a rapid, cheap and efficient method of polymerization.

A "seeding out" process has been developed by the chemists of the Bayer company. By it isoprene is heated in an autoclave, where spontaneous polymerization takes place. The process is said to yield a clear white rubber which can be vulcanized.

Vulcanization

By vulcanization we mean the change which takes place when rubber and sulphur are heated together, or when rubber is treated with a dilute solution of sulphur monochloride. The first is called hot vulcanizing and the latter cold curing, which is the word given to this change by rubber chemists. The early investigators of vulcanization called it simply the "change."

Hot Vulcanizing.—In 1839 Goodyear discovered that by treating mixtures of sulphur and rubber at high temperature, the range of temperature within which rubber retains its elastic and non-sticky properties was considerably increased over the crude form. He later also discovered ebonite, which is the end product of the reaction between sulphur and rubber.

Broadly speaking, normal soft vulcanized rubber contains from 2 to 4 per cent combined sulphur, that is, sulphur not extractable by solvents such as acetone. Hard rubber, or ebonite contains from 20 to 30 per cent combined sulphur. The maximum 32 per cent is seldom obtained in commercial ebonite, due to the extended length of time of heating and the large excess of sulphur necessary.

The action of sulphur with rubber seems to be partly physical and partly chemical. It is an addition process, either by adsorption or by straight chemical addition, or what is more probable, by both. It is not by substitution, as was originally thought.

The amount of sulphur which enters into combination with rubber, that is, the combined sulphur, increases with time and rise of temperature. The combination is not a linear function but shows certain jumps at more or less regular intervals. Of these jumps Weber says that they may be taken as probably indicating a particular physical rather than some definite chemical condition of the vulcanized product. His reason is that by curing at different temperatures the jumps occur at different places. Axelrod, on the basis of his experimental work, concluded that vulcanization consists of two separate processes. First, the action of heat on rubber which tends to de-polymerize it; and second, the entry of sulphur into the molecule, thus tending to increase the molecular complexity. He arrived at the conclusion that when de-polymerization due to heat has proceeded so far, the material is in favorable condition for a fresh sulphur attack, and at such points a jump occurs in the curve.

If temperature and time are kept constant, the amount of sulphur entering into the molecule is dependent on the quantity originally present. Weber noticed the fact that articles containing less than 2 to 2½ per cent combined sulphur did not appear vulcanized. Therefore, he concluded that this amount of sulphur is necessary to produce the first of a series of compounds called by him polyprene sulphides. This closely approximates the formula $(C_{10}H_{16})_{10}S_2$. By using varying large amounts of sulphur he obtained ebonites which all showed approximately 32 per cent combined sulphur, which would indicate another compound with the empirical formula $C_{10}H_{16}S_3$. He concluded that there exists a whole series of vulcanized bodies, of which $(C_{10}H_{16})_{10}S_1$ is the lowest, and $C_{10}H_{16}S_1$ is the highest.

Erdmann has brought forth rather an interesting idea. At 160 deg. C., which is practically the upper limit of vulcanization temperatures, melted sulphur contains a large proportion of thiozonide, S_8, molecules. Since thiozonide readily polymerizes to an eight-ring complex, it is highly probable that this ring would unite with another eight-ring such as rubber is supposed to be. This would give a rubber thiozonide corresponding to the well-known ozonides.

Wolfgang Ostwald has suggested that vulcanization is a physical proposition only, based on the adsorption of sulphur by the rubber colloid.

At present the consensus of opinion of American chemists seems to be that vulcanization is both a physical and chemical change, and that the adsorption of sulphur while actually taking place, is only a secondary matter.

Cold Vulcanizing.—In 1846 Alexander Parks patented the second method of securing the "change." He immersed his samples in a 2½ per cent solution of sulphur monochloride, S_2Cl_2, in carbon bisulphide. Of late carbon tetrachloride has been used instead of carbon bisulphide with equally good results, and with far less danger. The cold vulcanizing solution is called curing acid in the trade.

The process is only applicable to articles of less than 4 mm. thickness, since before the "acid" could penetrate to the interior, the surfaces would become hard and brittle, "burnt." The cold process finds considerable favor in the vulcanizing of thin goods, such as surgical gloves, etc. It is used quite extensively in combination with rubber cement, and to some extent in curing proofed articles, such as raincoats. The compound formed by cold vulcanization is chemical, and is believed to have the formula:

$$\left(C_{10}H_{16}\right)\begin{matrix}-S-S-\\ Cl\quad Cl\end{matrix}\left(C_{10}H_{16}\right)$$

A method of enameling iron, which may find application in the laboratory by reason of the simplicity of the procedure, is described by M. P. Wood in his work on Rustless Coatings. The metal is first pickled in hydrochloric acid to free it from foundry scale, then washed thoroughly and dried. The first coating applied is composed of 34 parts silica, 2 parts soda and 15 parts borax, mixed in water. The metal thus coated is exposed for 10 or 15 min. in a dull-red-hot retort. A second coating is then applied, consisting of 34 parts feldspar, 19 silica, 24 borax, 16 oxide of tin, 4 fluorspar, 9 soda and 3 saltpeter. This mixture is first melted in a crucible, then ground to a fine paste in a little water and applied with a brush. The coated piece is then again subjected to white heat in a muffle. It is claimed that the enamel unites with the iron and that pipes thus enameled have been in use for many years without deterioration.

Hydrometallurgy of Zinc and Lead in 1915

A Contribution from the Department of Metallurgical Research, University of Utah. D. A. LYON, Metallurgist in Charge. O. C. RALSTON and J. F. CULLEN, Assistant Metallurgists

The year 1915 witnessed the installation of a great number of semi-commercial plants for the treatment of zinc ores by leaching methods, with subsequent electrolysis of the solution in order to obtain metallic zinc. Practically all of these installations have been in the nature of experimental plants.

When the European war started there were three electrolytic zinc plants in Europe supposed to be operating on a commercial scale. All of these were on a small scale, and were producing zinc from by-products. At Winnington, Northwich, Cheshire, England, was the plant of Brunner, Mond & Co., which is an alkali works and which has been disposing of calcium chloride solution wastes by leaching zinc ores with them (with simultaneous action of carbon dioxide). This gave a zinc chloride solution from which the zinc was recovered by electrolysis, giving a spelter containing 99.96 per cent Zn., and producing chlorine gas which was absorbed to make bleaching powder. This process is one of those originally designed by Hoepfner, and was experimented on during the 90s in Germany, resulting in the installation of a like plant at Duisburg, Germany. Both of these plants have kept secret the exact details of the process by which they have been able to get good zinc deposits as their pure spelter has been for years at a premium of at least 1 cent above the ordinary market price.

In 1914, a plant was built for treatment of zinc carbonate ores at Kristiania, Norway, under the direction of Borchgrevink, and another at Balestrand, Sogn, Norway, under the direction of E. A. Ashcroft, to treat flotation concentrates from Broken Hill. So far as we know, nothing has been heard of these plants since the war began.

In the United States and Canada, however, a great deal of experimental work has been done on the leaching of zinc from ores with its subsequent electrolysis. Nearly all of this work has been sulphuric acid leaching followed by the electrolysis of the zinc sulphate solution. While all the various plants have this much in common, the details of the work varies at most of them by almost as many methods as there are plants.

The Anaconda Copper Company perhaps has gotten farther along with this work than have most of the other companies. They now have a 25-ton plant nearing completion. By a 25-ton plant is meant one that will produce 25 tons of metallic zinc per day. The Anaconda metallurgists regard this as only an experimental plant. Some splendid work has been done at Anaconda in their research work in connection with the development of the process which they are using and the publication of the details of this work by Mr. Laist and his associates will be keenly awaited by those interested in this line of work.

As to the general outline of the processes employed at the various plants at Trail, B. C., and at Anaconda, the electrolytic liquor containing sulphuric acid, regenerated by electrolysis with insoluble anodes, is used for leaching the ore. The ore is a roasted complex sulphide of lead and zinc carrying silver, some gold (and some copper in the case of Anaconda). By the use of barely sufficient acid to dissolve the zinc, it is possible to get a solution of zinc sulphate carrying low amounts of the other constituents of the ore. This solution must be cleaned by the use of zinc oxide, lime or other cleaning agents, and is then electrolyzed. At most of the plants, lead anodes are used in the electrolytic tanks in order to regenerate sulphuric acid. The process as thus outlined is old, but the refinement of detail has been the feature of nearly every plant attempting to use it.

At Murray, Utah, and Omaha, Neb., are small plants recovering zinc electrolytically. The one at Murray is following the practice as developed at Anaconda, with variations made necessary by local conditions. It is only an experimental plant and it is expected to produce about 2 tons of zinc per day. The Omaha plant is working on zinc oxide material from the refining of argentiferous lead by the Divine process, and is also in the nature of an experiment.

Perhaps the most extensive research work on the leaching and electrical precipitation of zinc has been carried out on the ores of the Bully Hill mine, Shasta County, California. The process for the Bully Hill ore was worked out on the assumption that a good deposit of zinc cannot be obtained from a solution containing much sulphuric acid and the effort has been to neutralize the acid as fast as it is formed at the lead anodes. In treating the Bully Hill ore, lime is used to precipitate zinc hydroxide and calcium sulphate from the solution of the zinc sulphate. This precipitate is suspended in the zinc sulphate liquor of the electrolytic cell and as fast as sulphuric acid forms it is neutralized by the zinc hydrate.

The Reed Zinc Company whose plant is at Palo Alto, Cal., has operated on the bag house dust from the Kennett smelter of the U. S. Smelting, Refining and Mining Company. This is also an experimental plant designed to produce about a ton of zinc per day. Instead of a solid lead anode they use a sponge lead anode resembling the electrode used in a storage cell. As fast as sulphuric acid is formed at this anode it combines with the lead, forming lead sulphate. This lead sulphate can later be used to give up the acid by reversing the current, after placing the electrode in a sulphuric acid solution. The solution for electrolysis is prepared by dissolving zinc sulphate crystallized from the leaching liquor.

At Silverton, B. C., the Standard Silver Lead Mining Company is also experimenting with the production of electrolytic zinc by the French process. French was probably the first man to use manganese in the electrolysis of zinc sulphate solutions, although its beneficial effect has been noticed by other investigators. He found that it deposits as dioxide on the anode and can be redissolved in sulphuric acid and used over again. He operates his process with considerable of the manganese sulphate in his electrolyte, and instead of using sulphuric acid for leaching, uses a solution of by-product sodium bisulphate.

At Welland, Ontario, the Weedon Mining Company is also said to be meeting with success in operating the Watts process. This process proposes the use of a solution of zinc sulphate for an electrolyte in electrolysis, the depolarization of the anode and the prevention of the formation of sulphuric acid by the use of any solid compound of zinc which does not contain objectionable impurities, such as zinc oxide, or blue powder. The fundamental idea of this process is the same as has been worked out in the treatment of the Bully Hill ore. It is also brought out in the recent patents of O. Best of Oakland, Cal.

At Mt. Read on the west coast of Tasmania, it has been reported that the Tasmanian Metals Extraction Company erected a plant for the treatment of zinc-bearing ores, producing a concentrate of zinc oxide. Nothing of the details of the process has been given out, but the zinc is doubtless precipitated from solution by a solution of some alkali, and the resulting hydrate calcined to oxide.

It is noticeable that most of these plants are treating complex ores of lead and zinc, and usually they are sulphide ores. There is a great amount of "complex sulphides" available in various mining districts and much of it has resisted all efforts at separation. Most of the lead smelters are receiving ores with a considerable zinc content and the zinc causes them to operate at a great handicap on account of the difficulty of fusing the zincky slag and due to the fact that a large amount of lead and silver is caused to go into the slag. Most of these smelters have been looking around for ways and means of removing this zinc before the ore goes into the blast furnace and there is no doubt that the removal of the zinc will allow much more economy in the lead smelting. For example, it is stated that one of the smelters in the country has concluded that even if zinc costs 7 cents per pound for removal and sells for 5 cents, the operation will still allow them latitude enough in the operation of the lead smelter to make the process slightly profitable. It is also stated that the Anaconda plant expects to produce zinc for 4 cents or even less. It is doubtful if small plants can produce zinc at such a low cost and it is also doubtful if it is possible at the present time to install a small electrolytic zinc plant which will be a paying proposition. In order to produce zinc cheaply electrolytically, extraordinarily cheap electric power is a very important item. Moreover, the plant must be comparatively large and this means that only the larger companies would at the present time be warranted in producing zinc in this manner, as this cost of installation is too great except for the large companies.

In other words, it now looks as if the production of zinc electrolytically is not a "poor man's proposition" for the reason that installation costs for a plant which will make money are liable to run well into the millions. However, as the work progresses, it may be found that this is not necessarily true.

The complex sulphides are, of course, being roasted before treatment, and it is worthy of note that most of the experimental work is being done with McDougal and Wedge furnaces. The roasting takes a considerably longer time than does the roasting in ordinary lead smelting and there is some question as to whether this type of furnace will do the work satisfactorily. Zinc smelters have used Hegeler and other like furnaces in which the time of roasting is as much as sixty hours, while most Wedge and McDougal furnaces are so constructed that the ore will travel through the furnaces in four hours, but can easily be slowed down to eight. Whether this type of roaster will meet the requirements or not is still a question.

Very little has been done with the carbonate ores which carry zinc, although it is understood that most of the smelters in Utah have experimented with the leaching of these ores. The Salt Lake station of the Bureau of Mines in connection with its work on the low-grade and complex ores has been doing work along this line and finds that most of the carbonate ores yield up their zinc readily. Very little has been done on the electrolytic deposition of the zinc in the solutions obtained by leaching these ores with sulphuric acid. The carbonate problem is not as pressing as is the mixed sulphide problem, for the reason that in most districts the sulphides constitute the main ore, and furthermore, the carbonates of lead which are found in most districts are not as seriously contaminated as are the zinc ores, for the reason that in the weathering process the lead carbonate resulting from galena does not travel far from the original galena while the zinc carbonates are often found at considerable distances from the original position of the sulphides. However, there are plenty of instances of mixed carbonates of lead and zinc that defy treatment, and being carbonates they have been let alone whenever it was found that they were complex, due to the fact that very few instances are known where they could be mechanically separated.

All things considered, it would seem as if a very decided advance had been made in the hydrometallurgy of zinc during the year 1915 and while the total amount of metallic zinc produced to date by electrical precipitation has been small, and at present commands a premium as "brass special," it will probably not continue to do so long, due to the large amount of it which will be produced by new plants now being installed or contemplated.

Lead

The hydrometallurgy of lead has received serious consideration during the past year. Inasmuch as it is sometimes found difficult to leach copper ores at a profit, it may seem strange that any one would even consider leaching an ore for such a low-priced metal as lead. However, if we compare the atomic weights and chemical equivalents of copper and lead, it will be seen that the quantity of chemicals required to extract a pound of copper should extract 3½ lb. of lead. Counting lead as worth 3 cents to the producer in the mountain region, the net result of the use of the chemicals which extract 1 lb. of copper will be 11 cents worth of lead. Although such a comparison looks favorable when made in this manner, the difficulty has been to find a suitable solvent for lead.

The Bureau of Mines station at Salt Lake City in its work on the recovery of lead values from the low grade lead ores made a study of the results obtained in three different mills during 1914 and found that the possibility of leaching lead was not at all discouraging. For example, at the mill of the Mines Operating Company, in Park City, the superintendent, Mr. Holt, found that his brine solutions built up to 8 lb. of lead per ton. He was giving his ore a chloridizing roast and then leaching with a 20 per cent salt solution. This was extracting the copper, gold and silver of his ore. Some of the lead precipitated on the scrap iron which was used for removal of the copper from the solution, but this was only a small part of the lead in his ore.

At the Knight-Christensen mill in Silver City, it was also noticed that lead was soluble in the brine solution and steps were being taken to recover this lead when the mill burned down. They had been very successful in recovering it on a laboratory scale by electrolysis of the solution.

The Bunker Hill and Sullivan mill at Kellogg, Idaho, had also been trying out a chloridizing process on some of the portions of their ore which had resisted concentration. The Malm process was used but was abandoned when it was found that there was not enough zinc in the ore to allow economical treatment of the ore in question, as the Malm process depends upon step precipitation of the various metals in solution until finally a solution of zinc chloride is left which is evaporated and electrolyzed. If the zinc is not present in the ore, it must be added.

During the experimental work on the Malm process, some observations had been made on the effect of brine solutions on chlorinated ore, which finally lead to the conclusion that most of the galena of the ore, even in an atmosphere of chlorine, was being converted largely to sulphate of lead, and that this sulphate of lead was soluble in a saturated brine. This was checked up by direct experiment and proved to be a most useful fact in the further development of a process. The brine had been used due to its known solvent power for silver chloride and lead chloride. Earlier in their work the Malm company had found that the solubility of lead

chloride in saturated solutions of salt is considerably increased over that in water, due doubtless to the formation of a double salt, and since they did their work, the solubility of the $PbCl_2$-NaCl system has been published in the *Compt. Rend.*, confirming their results. At the plant of the Bunker Hill and Sullivan company it was found that the lead had been extracted from the Bunker Hill and Sullivan ore due to the fact that the sulphate of lead was also soluble in the brine.

With these facts in mind, Mr. Handy of the Bunker Hill and Sullivan company decided to try roasting the ore with a small amount of salt, in order to chloridize the silver of the ore and allow the lead to either chloridize or to form sulphate as it pleased. This left the ore in a condition such that all of the lead and much of the silver was soluble in a saturated solution of common salt.

In co-operation with the Bunker Hill and Sullivan company, the Salt Lake station of the Bureau of Mines took up the investigation of this process at this point. Under the direction of C. L. Larson, who at that time was connected with the Metallurgical Research Department of the University of Utah, an investigation was made of the possibility of applying the Holt-Dern roaster to the chloridizing of the Bunker Hill and Sullivan ore. Exclusive of the salt, this roaster had shown a cost of 16 cents per ton for roasting, at the mill of the Mines Operating Company at Park City, Utah. The Bunker Hill and Sullivan ore only contained 4 per cent sulphur, but Larson was able to roast it without the addition of fuel, and at the same time he got good extractions of the lead upon leaching with brine. As to the details of the process, which has been developed by the Bunker Hill and Sullivan company, these will be given to the public at such a time as may suit the convenience of the company. In the meanwhile, an experimental plant is being operated to collect data on the best temperature of the solution to use and to work out other data necessary for installing the process on a commercial scale.

The Bureau of Mines is now attempting to apply this process, without roasting, to the carbonate ores of lead. In doing so, it has been found that most of such carbonate ores yield their values to a saturated brine solution containing enough sulphuric acid to convert the lead to soluble sulphate or chloride. Natural sulphates of lead, while somewhat rare, and always accompanied by carbonate, will dissolve in the neutral brine. On running in cycles it has been found that sulphate of sodium or any other sulphate does not accumulate in the solution but seems to stay behind in the ore, due to the fact that sodium sulphate is not very soluble in a saturated brine. Consequently, as many as thirty complete cycles have been run without deterioration of the solution.

Following up Christensen's work, it has been found that the electrolysis of brine gives a spongy lead. Unlike any other spongy metal deposited from solution, it is perfectly feasible to melt this sponge lead into bars without serious oxidation and even when it does oxidize to some extent a very small amount of reducing agent will reduce the lead with ease. Such a thing is not possible with copper or zinc sponge. Furthermore, with the use of an iron anode, iron will go into solution, thus lowering the operating voltage, so that with a current density of 69 amp. per square foot at the cathode, only half a volt is necessary, and the solution can be almost stripped of lead before the current density must be reduced. Under these conditions the almost unbelievable amount of 70 lb. of lead per kilowatt-hour can be removed from the solution as a sponge which settles into the bottom of the cell where it can be compacted and removed to be melted down. Furthermore, this spongy lead will not short-circuit the electrodes and hence the cells can be operated without much care. The iron going into the solution does not seem to affect the purity of the lead melted down and does not build up in the solution to much more than 1 per cent, some of it seemingly remaining in the ore.

The future of such a process is hard to predict but with such a great amount of mill solvent as the Great Salt Lake contains, it would look as though the low-grade carbonates of lead in Utah might be profitably treated by leaching instead of by furnacing. At least they could be concentrated into a 70-80 per cent product before going to the furnaces, and with a high extraction of the metal. Lead carbonates often have a bad tendency to slime and be lost in the tailing of a concentrating mill, but slimes need not interfere with a process of this kind. With several mills testing out the process, the developments in 1916 will doubtless be of considerable interest, both technically and commercially.

The Cost of Flotation

In a paper presented to the A. I. M. E. by J. M. Callow, the author gives the following data on the cost of flotation as practised by his pneumatic process, which was described in this journal Sept. 1, 1915:

Oil.—The oil mixtures generally in use cost from 1.25 cents per pound up to 3 cents per pound, depending on the percentage of creosol and other higher-priced oils used; on most ores 1½ cents per pound is a safe average figure, and a consumption of 1 to 1½ lb., or from 1.25 cents to 4.5 cents per ton of feed, averaging 2½ cents, would be a safe estimate.

Labor.—This will vary, of course, with the size of the plant. At one plant, consisting of 60 cells, two men per shift operate the entire plant, equivalent to a cost of 1¼ cents per ton. One man per shift on a 250-ton plant means a cost of 5.4 cents per ton maintenance. Assuming a life of three months per blanket, a capacity of 50 tons per cell, and an allowance for repairs to blowers, motors, pumps, etc., we have ½ cent per ton as a liberal allowance.

Power.—With power at 1 cent per kilowatt-hour, and a consumption of 2½ kw-hr. per ton, the cost would figure 2.5 cents per ton of feed.

Summarized, the estimated cost on a 2000-ton flotation plant, or larger, would be approximately as follows:

	Per Ton, Cents
Labor	1.25
Oil	2.50
Maintenance	0.50
Power	2.50
Total	6.75

On a plant of 250 tons the extra labor costs per ton would bring it up to approximately 10 cents per ton of flotation feed.

Actual figures from a large plant treating over 2000 tons by flotation gave 6.1 cents per ton; the flotation feed in this case represents 60 per cent of the crude ore tonnage, making the cost 3.5 cents per ton of crude ore treated.

Byproduct Coke Oven Plant.—The Indiana Coke & Gas Company of Terre Haute, Ind., has placed a contract with the Gas Machinery Company of Cleveland, Ohio, for a byproduct coke-oven plant, consisting of thirty cross regenerative byproduct coke ovens and apparatus for the recovery of tar and ammonia. Indiana coal will be used and the coke will be sold, for the time being, for domestic uses. The surplus gas will be delivered to the Terre Haute Gas Company for local consumption.

A Form for the Classification of Flotation Data

BY W. A. WHITAKER AND GEORGE BELCHIC

The increasing importance of flotation as a method for ore concentration and mineral conservation is illustrated by the fact that many private corporations, as well as university laboratories, are now engaged in investigating this process. Most of these investigators have as their object the defining of the conditions necessary for successful flotation when applied to particular material, while others are engaged in purely theoretical work looking toward an explanation of the various phenomena upon which flotation is based. Whatever the object sought, the investigation, if prosecuted to any extent, will result in the accumulation of a large amount of minute data, which unless arranged in such a way as to allow of ready comparison and correllation, may soon become a mass of unwieldy material, difficult of clear interpretation.

In order to compile in a condensed arrangement the results obtained in this laboratory and thus allow of quick and easy comparison and correllation, the form illustrated below was evolved and printed in quantity. With this arrangement, the investigator has before him on one sheet (1) the screen analysis, (2) the assays, (3) the charge used, (4) the derived data, and (5) the plotted efficiency of the frothing agent.

The same form may be used also when the effects of other flotation factors are being investigated, such as variations of temperature, duration and speed of agitation, quantity of acid, etc. This condensed scheme has worked so satisfactorily in this laboratory, it was thought that its publication might be helpful to those who are beginning investigations on flotation and possibly to those already launched upon the subject.

Division of State Chemical Research,
University of Kansas,
Lawrence, Kan.

Flotation Experiments

Number of Sample b Date May - 1915

Assay { Zinc - 5.17 %, Lead - trace } Number of Experiment 12

SCREEN ANALYSIS

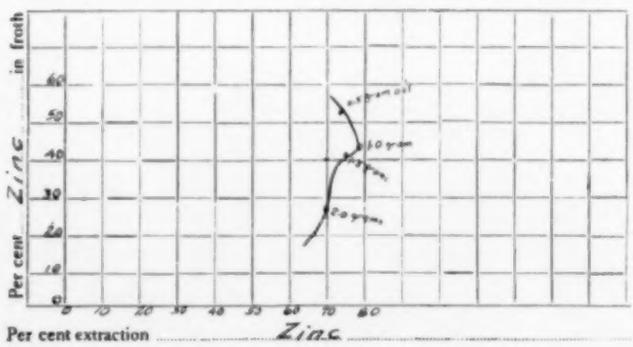
SCREEN			WEIGHTS			ASSAYS		
	Mesh	Opening Inches	Sample Weights grams	Per Cent.	Per Cent. Accumulative Weights	% Zinc		
All Pass.	35	.0164	—	—	—	—	—	
Retained	48	.0116	2.0	0.417	0.417			
"	65	.0082	3.0	0.625	1.042	2.54		
"	100	.0058	9.0	1.875	2.915			
"	150	.0041	34.0	7.08	9.995	3.8		
"	200	.0029	85.0	17.70	27.695	4.58		
Pass thru	200	.0029	347.0	72.30	—	5.56		
Totals			480.0	100.0				

To show the effect of variation in quantity Pine Oil

CHARGE		Variable grams	DERIVED DATA			
			Froth	Assay of Con.		% Ext
				in	Pb	
Ore, grams	400					
Water, "	1200	0.5	29.0	53.2	—	74.6
Acid, " H_2SO_4	2.0	1.0	38.0	43.1	—	79.2
Oil, "	Variable	1.5	37.0	42.4	—	75.8
Temp. in °C	65	2.0	51.0	28.6	—	70.0
Speed, RPM	2000					
Time, minutes	7					

EFFICIENCY CURVE

Efficiency of Pine Oil in terms of frothy assay and extraction.



Per cent extraction Zinc

Reverberatory Smelting at Consolidated Arizona Smelting Company, Humboldt, Arizona

(Editorial Correspondence)

In a previous article in this journal[1] we described in detail the concentration of copper ores by flotation at Humbolt, Ariz., and gave many valuable data on the method and cost of operation. In conjunction with the concentrator, the company operates a smelting plant for the treatment of its own ores and concentrates and certain ores bought on a custom basis. The custom business is being built up as rapidly as the development of the district warrants. Smelting is done in oil-fired reverberatory furnaces, and copper is produced from matte in basic-lined converters. Production has averaged about 500,000 lb. of blister copper per month, two-thirds to three-fourths of which is from company ore and the balance from custom material. Costs of operation and production will be officially announced by the company in the near future.

The equipment of the plant consists of one seven-hearth Wedge roaster, 21 ft. 6 in. diameter, fitted with auxiliary oil burners; one 19-ft. by 60-ft. and one 19-ft. by 100-ft. reverberatory furnaces, the latter being under construction and practically completed at the time of writing, and two 9-ft. 6-in. diameter barrel-type basic-lined converters.

Table I gives in condensed form average analyses of some of the materials treated and products made. Gold and silver are indicated in ounces per ton and the other constituents in per cent.

TABLE 1.

	Au.	Ag.	Cu.	Insol.	Fe.	CaO.	S.
General mill conc...	0.09	3.04	9.52	16.5	30.0	..	...
Smelting ore	0.04	1.50	3.70	46.0	20.0	1.3	22.0
Roaster feed	...	...	6.00	20.0	30.0	1.4	20.0
Calcine	...	...	6.20	23.5	35.5	1.4	9.0
Reverberatory slag.	...	...	0.35	38.0	38.0	0.8	...
Matte	...	...	35.00	...	...	..	...
Blister copper.....	1.00	28.00	99.10	...	...	..	...

In addition to the company's own products, a considerable tonnage of custom ore is treated, a large portion of this being a copper-bearing specular hematite containing 48 per cent iron and 10 per cent insoluble. Converter flux and fettling material is highly silicious copper ore from the upper levels of the company's mines, and contains 3.8 per cent copper and from 70 per cent to 80 per cent insoluble. In this ore the copper minerals are more or less oxidized and not well suited to concentration, particularly flotation, tending to "kill" the froth in the latter process.

It will be noted that all material entering the plant at any stage of the process is copper-bearing. Barren material is excluded, and by this practice every flux adds

[1] December 1, 1915, p. 897.

its quota of copper while serving the purpose of producing the desired slag. In this respect the practice under the present management is an economic improvement over that of an earlier period when blast-furnace smelting was practiced and barren fluxes were purchased.

Roasting Flotation Concentrates

Roaster feed is a mixture containing smelting ore, general mill concentrate and copper-bearing iron oxide. Due to the fact that about 75 per cent of the general mill concentrate is a flotation product, 60 per cent of which is finer than 200-mesh, the general concentrate contains from 40 per cent to 45 per cent of material of that mesh and finer. The smelting ore and iron oxide are crushed to only ½-in., but even so, the roaster feed still contains a large percentage of very fine material. This naturally gives rise to excessive dusting in the roaster, demanding special attention to dust recovery in order to avoid a serious loss.

Roaster capacity with this feed is 110 tons per day. The sulphur content of the feed is maintained at about 20 per cent, and as long as it is 17 per cent or more there is no need of applying extraneous heat to carry on the roasting process. Oil burners are provided at the hearth doors, however, for use in case of necessity.

Typical roaster feed is composed of the following constituents by weight, and has the general analysis given in Table I:

Smelting ore	600 lb.
Specular iron ore	3,400 lb.
Mill concentrate	6,000 lb.

To this mixture, in the form of calcine, is added the following cold material to make up the reverberatory charge, the total weight being the charge per hour and six minutes, fed at thirty to thirty-five minute intervals:

Converter slag	1,500 lb.
Specular iron ore	2,500 lb.
Mill concentrate	1,500 lb.

It will be apparent that, when fettling material is included, an unusually large part of the total furnace charge is cold material, amounting roughly to 50 per cent. The conditions are quite different to those prevailing in plants where the reverberatory charge is practically all hot calcine and fluid converter slag. This accounts for the comparatively high fuel ratio, which is one barrel of crude oil per ton of total charge. The oil is preheated to a temperature of 160 deg. Fahr. and is fed to the burners under 70-lb. pressure. The furnace is equipped with two 350-hp. Stirling waste-heat boilers, but no very positive data are available from which to calculate a credit to oil consumption represented in heat recovery at the boilers. Under the conditions imposed this 19-ft. by 60-ft. reverberatory furnace has a daily capacity of 165 to 175 tons. The average grade of furnace charge and products is shown in Table 1.

The new reverberatory, 19 ft. by 100 ft., is constructed with a continuous hopper across the bridge and along the side walls. Fettling will be done with a blended neutral mixture through pipes from the side hoppers instead of by hand in the old manner. Due to the character of the material to be smelted, the charge for this new furnace will be temporarily on a basis of 30 per cent calcine and 70 per cent cold charge, the latter being in even greater ratio than in the smaller furnace at the present time, although it is probable that this will be modified by providing additional roaster capacity at a later date.

Intermittent Use of Basic-Lined Converters

Converter capacity is far in excess of requirements when only the small reverberatory is in operation. This results in the intermittent use of a shell, and the practice here has thrown valuable light on the use of basic-lined converters under such conditions. The question has previously been raised as to whether converters of standard size lined with basic brick could be successfully used in comparatively small plants where intermittent operation would be necessary on account of the small output of matter. It had been feared that the converter would be subjected to such severe treatment due to the expansion and contraction of the brick lining that its use under these conditions would be unprofitable. This question was raised in the copper symposium at the International Engineering Congress in San Francisco, and at that time Mr. Laist stated that the basic-lined converter had been found to stand hard usage in intermittent service, with little cracking of the lining.

Experience at the Humboldt smelter confirms this fact, for under the method of operation no serious difficulty has been encountered. When a new lining is put in service, a number of charges are blown at short intervals to saturate and bond the brick with metal. After the lining is thus seasoned the converted will be idle for seven to eight hours between blows. On occasions a shell has been idle as long as thirty hours after a blow. According to conditions and the probable length of time the converter is likely to be down, the lining is kept warm by adding a charge of fuel, but otherwise no provision is made for the care or protection of the lining. No data are available as to production per converter lining, as the converters have not been in operation for a sufficient length of time to get any direct figures.

While magnesite brick has proved satisfactory under these conditions, Mr. E. C. King, smelter superintendent, plans to try a monolithic lining of magnesite bonded with silicate of soda or magnesium sulphate. This will be done with the idea of securing a less expensive lining, and at the same time one which may show an improvement over the use of brick.

With the new 100-ft. reverberatory in use, the smelting capacity of the plant should be increased to 450 tons per day, with a monthly production of about 1,000,000 lb. blister copper. At present the company is not equipped for blast-furnace smelting, but the erection of a blast furnace is in prospect if mining conditions in the district develop as anticipated so that a supply of suitable ore can be assured. With the ores just now available blast-furnace smelting is out of the question as a matter of economy and good metallurgy.

Our thanks are due Mr. Colvocoresses, general manager, and Mr. King, smelter superintendent, for their courtesy in favoring us with the foregoing information.

Mesothorium, a by-product from the manufacture of thorium nitrate from monazite, has properties similar to those of radium. It is considered possible that this country's resources of monazite can be used profitably in the manufacture of this valuable material. The domestic supply of monazite is found mainly in North and South Carolina, Idaho, and in the black sands of the Pacific slope.

New Regulations Governing Export Procedure.—The announcement that new United States regulations relative to export procedure will become effective Jan. 1, 1916, has created such intense interest among manufacturers and shippers that the Bureau of Foreign and Domestic Commerce, Department of Commerce, has found it necessary to reprint the new order with explanatory text. That pamphlet is just off the press and is being supplied free of charge to those interested, upon application to the above-mentioned office. All shipments for export to foreign countries or to Alaska, Hawaii and Porto Rico will be affected by the new regulations.

The Central Mill of the North Star Mines Co.

BY LEROY A. PALMER

In the Sept. 15 issue of METALLURGICAL AND CHEMICAL ENGINEERING I discussed two mills in the California gold belt, one practicing amalgamation and concentration and the other straight cyanidation. This article will deal with a mill that uses all three processes. Mills that amalgamate and then either concentrate or cyanide the tailings are common, but the process in vogue with the North Star Mines Company is unique in that following amalgamation the tailing is concentrated and the concentrate subsequently cyanided with the concentration tailing.

The North Star Mines Company operates three mills, the Champion at Nevada City and the Central and North Star at Grass Valley, Nevada County, Cal. The ore carries both gold and silver in a quartz gangue, the precious metals occurring both free and associated with pyrite, galena and sphalerite. The same process obtains at all of the mills of this company and the following on the Central mill may be taken as generally descriptive.

The ore is hoisted from the Central shaft in automatic dump skips to the bins, from which it is hauled in electric trains with side dump cars to both the North Star and Central mills, the latter being only 200 ft. distant from the head-frame. The mill, Fig. 1, which has a nominal rating of 160 tons daily, is of timber frame with corrugated iron sheathing and concrete floors, and so set on a hillside that all conveyance of ore is by gravity.

Wet Crushing

The ore is dumped into two bins, each of 200 tons capacity, fitted with 1½-in. grizzlies. The oversize of each grizzly goes to a 9-in. by 15-in. Blake crusher set to 1½ in. These crushers are run wet and discharge to that portion of the bin beneath the grizzly. A 15-hp. motor drives both crushers at a speed of 180 r.p.m. The ore is fed to them from rack and pinion gates through a steel chute fitted with vertical grizzly bars which can be dropped across it to check the flow of ore and allow waste rock to be sorted out. In this way about 5.5 per cent of the ore sent to the mill bins is rejected without going through the crushers.

Masonry Mortar Block and Steel Battery Posts

The mill has forty stamps driven as four batteries by a 100-hp. motor, but working in five-stamp mortars, Fig. 2. The stamps weigh 1050 lb. and make 107 7-in. drops per minute in order 1-5-2-4-3, with 6-in. discharge. The feed is from Challenge feeders driven from the middle stem of each five stamps. Stamping is done through a steel screen with punched round holes 1/25 in. in diameter. At intervals of 1 in., laterally and vertically, a strip 1/16 in. wide is left unpunched for reinforcement. Such a screen gives an open area of 0.362 sq. in. per square inch of surface and has an average life of twenty days with an issue of 20 tons per day.

There are many points of interest about the batteries at the Central. First is the masonry foundation. When the mill was built in 1904 most designers clung to the old wooden battery-block. Masonry or concrete was looked upon with suspicion as it was considered that the mortar foundations must possess a certain amount of resiliency. The foundations at the Central are of granite bound with cement mortar, with concrete tops in which a rubber bedplate 1/32 in. thick is set so as to make a water-tight joint between the mortar and its foundation. The tunnel which frequently forms part of a battery foundation to permit tightening the nuts of the anchor bolts has been omitted in this construction, and the bolts are solidly embedded in the masonry. This practice has been very satisfactory as there has been no breakage of bolts or mortars and loose nuts are of rare occurrence.

Another feature is the steel battery posts at which so many millmen are inclined to look askance. Each post is made of two pairs of 8-in. channel irons bolted to the foundation and with a 4-in. spreader between the channels of each pair. This style of frame has been in continuous use since the mill commenced operation, with the failure of only four posts. When breakage has occurred it has been of a single channel iron at a time. In such a case the camshaft bearing is jacked up and the broken channel removed and replaced without stopping the battery. The advantage of this rigid construction has been well exemplified at the North Star mill of this company. With wooden battery frames the breakage of camshafts ran as high as eleven in a single year. At this latter plant cast-iron battery frames were substituted about a year ago and, as a result, the breakage reduced to four in that time, a favorable showing considering the weight of the stamps and the speed at which they are run.

The screen frame used is out of the ordinary. It is a double frame, one piece fitting inside the other and bearing against a face 1 in. wide around the inside of the outer frame. Instead of being tacked on, the screen is laid in the outer frame against a rubber gasket; the inner frame, which is of cast iron to obviate swelling, is laid against it and secured by three iron buttons. In fact, the screen is held in its frame in practically the same manner that the glass is held in a picture frame. It can readily be seen that this method

FIG. 1—CENTRAL STAMP MILL IN BACKGROUND, CYANIDE PLANT IN FOREGROUND

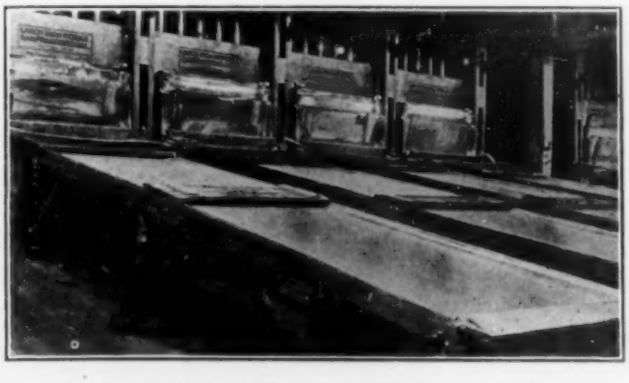


FIG. 2—CENTRAL MILL—BATTERIES AND PLATES SHOWING STEEL BATTERY POSTS

effects a considerable saving of time in setting up screens as compared with tacking them to their frames.

In place of the ordinary chuck-block for inside amalgamation, riffle-blocks with a heighth of 6 in. are used. Lengthwise of these blocks are stapled six rods of ¼-in. square iron which act as riffles to catch and retain the amalgam. This arrangement permits of running the battery with lower discharge, and consequently faster crushing and less sliming, than with the regular chuck-block. The amalgam is easily cleaned from the riffle-blocks, most of it dropping off by simply tapping the block when it is removed from the mortar and the remainder scaling off readily with a sharp instrument such as the point of a taper file.

Stamp stems are of uniform thickness throughout instead of being tapered at the ends to fit in the boss-heads. In place of the tapering end a two-part tapered bushing of ½-in. steel is used. This is set in the hole in the boss-head with the straight stamp stem inside it. In case a stem breaks, instead of taking it out of the guides and turning it around, the bushing and broken part are driven out of the boss-head by means of a drift, the stem is reset in the bushing without reversing, the tappet is set up and the stamp is ready for operation.

New Design of Cam

The latest innovation is the new cam designed by Arthur B. Foote, general superintendent, which is going into use in these mills and is illustrated in Fig. 3. It was found that with the speed at which these stamps

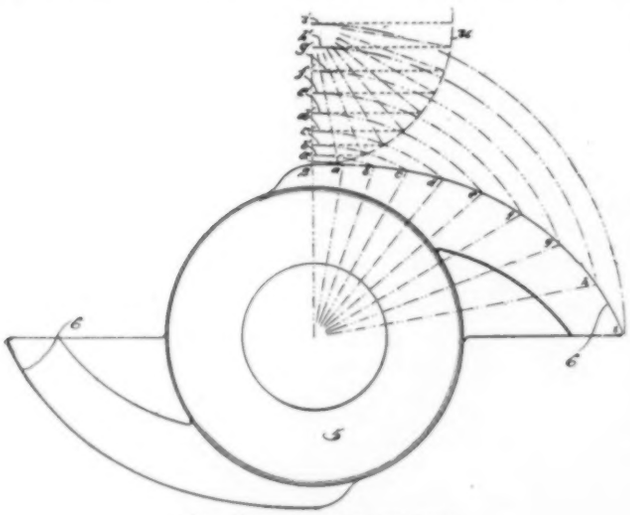


FIG. 3—CAM FOR STAMPS

were dropped, cams and shafts were frequently broken. Aside from torsion, to which any shaft is subjected and which is greater than usual in a camshaft due to the weight lifted, a camshaft is necessarily subjected to two strains. These strains are due to the impact of the cam engaging the tappet and the jar of the stamp transmitted to the shaft through the mortar, block and battery frame. If the mortar and block are well built and the battery frame rigid, the latter strain is negligible and the most serious effect on the shaft is due to the jar of the cams picking up the tappet, which this new cam is designed to reduce to a minimum.

As is well known, the cam which has been in universal use for years past is laid out on the slightly modified involute of a circle. This new cam can best be understood by referring to the accompanying illustration. The distance from the center of the shaft to *i'* is equal to the major radius of the cam. From *s* to *i'* is the throw. With *i'* as a center and *i's* as a radius, a quadrant is struck and divided into nine equal parts, each subtending an angle of 10 deg. Horizontal lines drawn from these points on the arc divide the vertical radius into parts, *sa'*, *sb'*, *sc'*, etc., representing the sines of 10 deg., 20 deg., 30 deg., etc., respectively. A similar quadrant with a radius equal to the major radius of the cam is struck from the center of the shaft and likewise divided into 10-deg. arcs.

The curve is then laid off by striking an arc from the center of the shaft, the radius of which is equal to the distance from this center to *a'* until it intersects the radius *a* of the major arc, thus fixing the first point of the curve. Subsequent points are fixed by striking an arc from the center to *b'* until it intersects radius *b;* from the center to *c'* to intersect radius *c*, and so on. The effect is a curve which may be likened to the spiral in a railway track, giving a radius which increases uniformly and thus imparts to the cam a uniform acceleration of speed such that when it finally disengages the tappet the velocity of the stamp is sufficient to carry it still farther upward and give it a drop greater than the actual throw of the cam.

It is claimed for this cam that it picks up the stamp with less jar than the involute design, a point that can be demonstrated by feeling the tappets of the two in operation. It requires special design for different drops as the blow on the tappet increases as the drop is shortened until at one-half the drop for which the cam is designed the shock of cam and tappet engaging would be the same as with an involute cam. Wear on the tappets is greater than with the involute design but breakage of cams and shafts, much the greater evil, is less. The designer makes no claims for it as a cure-all for battery troubles and recommends it only where there has been trouble from breakage due to high speed.

High Recovery by Amalgamation

From the batteries the pulp flows over eight amalgamation plates each 3 ft. by 20 ft., supported on frames made of angle irons, with a grade of 2¼ in. per foot. The recovery by amalgamation is about 79 per cent, an unusual amount on California ores. The riffle-blocks are cleaned every fifteen days and yield about 65 per cent of the free gold recovered. The battery plates are cleaned daily and yield most of the remainder. Traps are provided below the plates and small plates below the tube-mill, but the recovery from these sources represents only a small percentage of the total. About 100 to 130 oz. of mercury is charged daily.

On clean-up days one of the mortars is fitted with a 30-mesh screen and all sands from the clean-up and all "specimen" ore are run through this battery. The tailing from this treatment and the sand left around the dies when the battery is opened are collected in a tank and returned to the mill bin for retreatment. Sand from this clean-up is treated in a bowl, the overflow of which is settled in a concrete sump.

Concentration

After passing the traps the tailings from each of the eight plates go to a Dodd buddle, Fig. 4. This is an outward-sloping, circular-deck table, 8 ft. in diameter, with linoleum cover and annular riffles. It is driven by a head motion in which the action is transmitted through a crank with a speed of 190 r.p.m. and a throw of 1 in., so connected to the deck that the latter moves forward and back in the arc of a circle. The riffles are spaced 1½ in. apart, and stratify the pulp, which is fed at the center, on the same principle as the quadrilateral deck tables. Beneath the deck of each table is a small spiral pump which receives the middling and discharges it through a hollow central shaft. These tables make concentrates, middlings and tail-

FIG. 4—CENTRAL MILL—DODD BUDDLES

ings. The concentrates go to a tube-mill for regrinding, the tailings are laundered to the cyanide plant and the middlings are sent by their individual pumps to the feed box of a ninth buddle, which makes the same products but retreats its own middling.

Tube-Milling of Concentrates

The concentrates from the buddles go to a 2½-in. centrifugal pump, which elevates them to a 24-in. dewatering cone. The overflow of this cone goes to the tail race and the underflow to a 5-ft. x 10-ft. tube mill which makes 14 r.p.m. The tube uses Danish pebbles and crushes so that 97 per cent of its product will pass a 200-mesh screen. Because of the nature of the feed both pebble and power consumption are low, a 20-hp. motor driving the tube mill and the nine buddles.

The tube-mill product is run over a 3-ft. x 6-ft. amalgamating plate, and thence to a 24-in. cone. The overflow of the cone goes to the cyanide plant and the underflow is returned to the tube mill by way of the centrifugal pump.

Thus the sulphides, after being separated from the gangue as concentrates, are crushed and again mixed with it, prior to cyanidation. The sole object in the concentration is to reduce the amount of fine grinding necessary in preparing the plate tailings for further treatment, and in actual practice the buddles may be said to act as classifiers for the tube mill, the classification being gravimetric instead of volumetric.

It has been found that such precious metal as can be recovered from the concentration tailing by cyaniding is amenable to this treatment without further crushing than that given in the batteries. On the other hand, the gold occurring with the sulphides is intimately associated with them and fine grinding is necessary to free it sufficiently for the solutions to act on it.

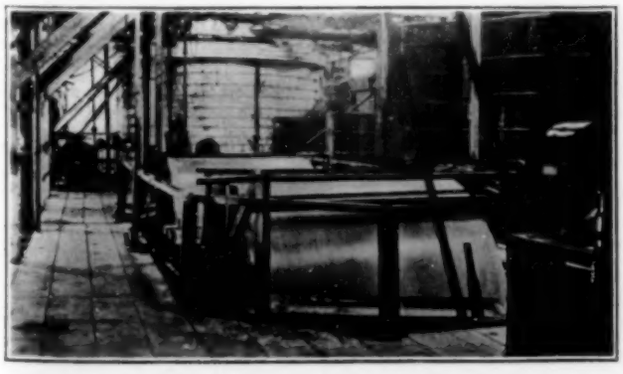


FIG. 5—OLIVER FILTERS, PACHUCA AND DECANTATION TANKS

As concentration is effected at a ratio of 40 to 1 this separation eliminates 97½ per cent of the tonnage from tube milling. The saving in power and pebble consumption is even greater than this, as the tailings are all quartz and the concentrates, the tube-mill feed, being sulphides, are much softer and more friable.

Cyanidation of Sand and Slime

The cyanide plant is situated about 500 ft. from the stamp mill at such lower elevation that the feed may be delivered at the top floor by gravity. The mingled tailings and concentrates, which classify as about 60 per cent slime and 40 per cent sand, are delivered to a 5-ft. sloughing-off cone. The underflow of this cone goes to two more 5-ft. cones, the overflow of which is divided between two 5-ft. x 6-ft. *V* settlers. The underflow of the second cone and the settlers goes by means of a Butters distributor to a leaching vat, of which there are six, each with a capacity of 125 tons.

Slime Treatment

The overflow of the various settlers mingles and is distributed by launders to six cone-bottom settling tanks, each 10 ft. x 12 ft. and to one Dorr thickener. The settler overflow is used as wash water on the leaching vats. A preliminary charge of lime is added at the settlers and the Dorr thickener, and the pulp drawn off at sp. gr. 1.30 to a stock tank in which sufficient lime is added to bring the strength up to 0.01 per cent; and cyanide of sodium to make the strength of the solution 0.03 per cent. This is equivalent to 0.0375 per cent, or 0.75 lb. KCN per ton of solution.

The pulp in the stock tank is agitated by compressed air and transferred by a Pohlé lift to the first of three Pachuca tanks, each 8 ft. by 15 ft., through which it is passed in series. From the last Pachuca the pulp is divided between two decantation tanks, the underflow from which goes to two Oliver filters and the overflow to the filter-solution sump. The Oliver filters make a revolution in 4 min. 20 sec., and receive a barren weak solution as a wash. The pulp is then wasted.

Details of Sand Treatment

After a leaching vat has filled and drained, the first treatment is by pumping on pregnant solution from the Oliver filter and the decantation tanks. These solutions are collected in a 450-gal. sump and raised by a 4-in. pump to the vat where they are brought up to a standard of 0.10 per cent NaCN. Thirty tons are thus pumped onto the sand and allowed to drain through. Thirty tons of strong solution, 0.10 per cent NaCN, are then run on and allowed to stand twenty-four hours. At the expiration of this time the valves are opened, and three hours after sand shows, filter solution is again run on until the sand is covered, when the vats are allowed to drain completely. The filter solution being drained, seven 15-ton charges of weak solution, under 0.06 per cent NaCN, are pumped from the sump below the Merrill presses, the vats being aërated by draining between charges until the sand appears. With the final draining of the weak solution the sand is washed for six hours before discharging with the lime water which overflows from the slime settlers.

When a vat is ready for discharging the valves, which are of Butters type, are raised, and a hose with a long nozzle thrust down beside each valve to start the sand out. When a channel is once made a Butters distributor is swung over the vat, a good head of water turned on and no further attention required except to sluice out a little sand which clings to the sides and bottom and which the distributor fails to dislodge. In this manner

a 125-ton vat can be emptied in about two hours' time with practically no personal attention.

Precipitation by Zinc Dust

The effluents of the leaching vats are titrated and sent to the pregnant strong or pregnant weak sumps, according as they are over or under 0.06 per cent NaCN. From these sumps they are pumped, after the addition of zinc dust, to one of two 10-frame, 30-in. Merrill filter presses. The precipitate is fluxed and melted without previous roasting or acid treatment.

Consumption of Chemicals and Power

The stamp mill uses about 0.75 oz. mercury per ton of ore treated. An average for a year in the cyanide plant showed consumption of chemicals per ton as follows: Sodium cyanide, 0.502 lb.; lime, 3.50 lb.; zinc dust, 0.192 lb.

Two motors are used in the cyanide plant, a 10-hp. belted to a 16 by 10-in. Rix single-stage compressor, which furnishes air at 15 lb. for the Oliver filter, Pohlé lift and Pachuca tanks, and a 20-hp. motor belted to a counter shaft, which drives all of the other machinery, including a No. 4 Roots cycloidal blower, which produces vacuum at 20 in. on the Oliver filter.

The following is a summary of all motors:

Machines	Motors	Horsepower
2 Blake crushers	1	15
40 Stamps	1	100
8 Buddles 1 Tube mill 1 Centrifugal pump	1	20
1 Air compressor	1	10
3 Pumps 1 Root's blower	1	20
Total	5	165

This indicates a consumption of 1.03 hp. per ton per day. A segregation of power costs for an average month shows cost per ton of ore as follows:

Crushing and stamping	$0.126
Concentrating and tube-milling	.016
Cyaniding	.036
Total	$0.178

Power is purchased from the Pacific Gas & Electric Company, which has extensive hydro-electric developments in this vicinity. The main transmission is at 40,000 volts; it is stepped down to 4000 for transmission to the mine, and there stepped down to 440 for use in the mills.

Extraction and Costs

An average value of the ore milled from the North Star mines would be very close to $10 per ton. Extraction is at the rate of 97 per cent, leaving a tailing loss of 30 cents per ton. Costs of operation for the year 1914 were as follows:

Crushing, stamping and amalgamating	$0.423
Concentrating and tube milling	.117
Cyaniding	.396
Total	$0.936

Most of the mills in California, especially the smaller plants, treat their ores by amalgamation and concentration, and ship the concentrates to the smelters. This practice was followed by the North Star for some time so that some comparison of the two methods can be made. Seventy-nine per cent, or $7.90 per ton, is recovered by amalgamation. Thirty cents is lost in the tailings, so that $1.80 is recovered by cyanidation. This costs $0.396 per ton plus the cost of tube milling, which will approximate 6 cents a ton. Treasure freight and mint charges amount to $0.0075 per ton of ore treated, so that this charge on the cyanide bullion would amount to $0.0014 per ton of ore, giving a total of $0.46 for all charges made necessary by cessation of shipment of concentrates. This leaves a net profit of $1.34 per ton from the plate tailings.

At a new mill on the Mother Lode in which a somewhat similar ore is concentrated the recovery on the vanners is 72 per cent. Applying these figures to the North Star the recovery from concentration would be $1.51 per ton. The smelter paid for 92 per cent of the gold at the rate of $19.50 per oz., so that the amount paid by the smelter for the concentrates from a ton of ore would be $1.31. Freight and treatment charges amount to $10 per ton of concentrates, or $0.25 per ton of ore with a concentration ratio of 40 to 1. Thus the net return per ton of ore concentrated would be $1.06, a difference in favor of cyaniding of $0.28 per ton.

I am indebted to Mr. Arthur B. Foote, general superintendent, and Mr. Frank Provis, foreman of the Central Mill, as well as other employees of the North Star Mining Company.

San Francisco, Cal.

The Mechanical Principles of the Blast Furnace

BY J. E. JOHNSON, JR.

The opinion that the mechanical principles of the blast furnace are insignificant in number and importance has been quite generally held, and the subject has been correspondingly neglected, but in my judgment this view is founded on superficial observations, since the proper mechanical action of the furnace is the key to its successful commercial operation. There are two main requirements of mechanical action of equal importance since the fulfillment of both is vital to the process. These are: first, that the stock must settle uniformly over the whole area of the furnace and at as regular a rate as possible; second, that the gas formed by the combustion in the hearth must rise with similar uniformity of distribution and velocity so that each portion of the charge shall receive its share of the action of the gas.

In the early days of blast furnace operation there was little or no conflict between these two requirements, but with the increase in the rate of driving previously described they have become to a certain extent conflicting because of the resistance to the descent of the stock column offered by the ascending current of gas. The gas is driven through the furnace by mechanical power, and the amount of pressure which we can put upon it is in recent years virtually unlimited, so that the gas (blast) pressure has risen as fast as there was any need for it to do so; in some cases, one might almost say, faster. But the force causing the descent of the charge column is simply that of gravity, which we cannot increase except by increasing the height of the furnace, and the net result of that remedy, as I shall presently show, is to make conditions much worse instead of better.

Our first care must obviously be so to distribute the charge in the top of the furnace that the resistance of all portions of the charge column shall be the same, as already pointed out in the chapter on filling the furnace. We know but little about the theory of stock distribution further than the necessity of placing a considerable part of the fine material against the walls on account of the greater proportion of voids at the surface of the wall. Under the head of operation we shall discuss this a little more fully, but there is not enough known on the subject to be of use in this discussion, further than the above.

The Blast Pressure

Within quite recent years, and perhaps even now, there is a very considerable misapprehension on the

subject of the cause and function of the blast pressure. This is made up of three parts, the pressure required to drive the air through the tuyeres, that required to force it up through the charge column, and that to force it from the top of the furnace out through the down comer and flues. These three items make up the total blast pressure just back of the tuyeres, while to obtain the engine house pressure the pressure due the resistance of the hot blast main, the stoves, and the cold blast main must be added, the sum of which should not exceed ½ lb. in good practice.

Many people have thought that the greater portion of the whole pressure was expended in driving the blast through the tuyeres, and that the resistance of the stock column was almost a negligible quantity, but nothing could be further from the facts. The pressure required to drive the blast through the tuyeres is in ordinary practice only about 10 per cent of the total pressure, and almost all the rest is expended in overcoming the resistance of the stock column. Finally from 1 to 3 per cent of the total is required to drive the gas through the down comer flues and burners.

In a paper published by me before the American Institute of Mining Engineers in 1905, entitled "Notes on the Physical Action of the Blast Furnace," there was given a discussion of this subject which, in the light of ten years' further experience, does not seem to require much modification, and is quoted here.

The Top Pressure

This is so simple that misconception would have seemed impossible had the attempt not been made to deduce results as to the quantity of flue-dust carried over at different furnaces from their top-pressures. The truth is that the top-pressure depends on the volume and density of the top-gases, and on the frictional resistance of the flues or mains through which they are forced, and on nothing else. The frictional resistance depends on the clear area, the length and the roughness of the mains.

Considering that the condition of the latter is generally a matter of entire uncertainty, as they vary in normal operation from perfectly clean to almost completely blocked with flue-dust and coke, calculations of the frictional resistance during operation are entirely worthless, as are also deductions as to the furnace work from the gas-pressure. If the top-pressure rises, the blowing engine automatically blows a little more pressure (never in good practice over ½ lb.), so as to maintain the same difference between the pressure at the tuyeres and that at the top. As these variations are much smaller and much slower than those due to other causes, they are naturally never being noticed at the engine.

The Pressure Required for the Tuyeres Penetration

The pressure drop through the tuyeres has also come in for its share of misunderstanding. Within a few years the assistant to the superintendent (now superintendent) of the largest blast-furnace plant in the world contended with me strongly that the blast-pressure was nearly all expended in forcing the blast through the tuyeres, and that the pressure at the top of the bosh would not be over 2 or 3 lb. in a furnace driven with about 15 lb. He was at that time making experiments bearing on this subject, and soon found his error. This is, in fact, a matter on which we can make calculation with certainty, Flieguer, Zeuner and Weisbach having made experiments on the flow of air through orifices, and established formulæ therefor which are authoritative.

The formula for this case is

$$W = 63.6A\sqrt{\frac{P_2(P_1 - P_2)}{T_1}} \qquad (1)$$

W being the weight in pounds of air passing each tuyere per minute; A, the area of the tuyere; P_1, the absolute pressure outside the tuyere; P_2, the absolute pressure in the furnace; and T_1, the absolute blast temperature (= deg. Fahr. + 460).

One thousand cubic feet of air at normal pressure and 70 deg. Fahr. weigh exactly 75 lb. Since this is about an average normal condition, the conversion of volume to weight can be conveniently made on this basis (and is so made throughout this paper). Moreover, since the tuyeres are usually circular in section, we can convert the above formula to a more convenient form for immediate use, as follows:

$$P_1 - P_2 = 2.4\left(\frac{T_1 V^2}{P_1 d^4}\right)$$

where V^2 is the volume of blast (free air) passing each tuyere per minute, measured in thousands of cubic feet, and d is the diameter of the tuyere in inches. This formula is not quite exact, being too high for low tuyere velocities and too low for high one; but it is accurate enough for the present purpose, under ordinary conditions.

For a furnace blown with 40,000 cu. ft. of blast per minute at 15.20 lb. pressure and 1000 deg. Fahr. temperature, through twelve 6-in. tuyeres, we have $P_1 - P_2$ = difference of pressure passing through the tuyeres =

$$2.4 \times \frac{1000 + 460}{15.2 \times 14.7} \times \frac{(3.3)^2}{6^4} = 1.0 \text{ lb.}$$

This is the difference in pressure required to give the necessary velocity through the tuyeres, and is, of course, fixed by the temperature, pressure, and volume of the blast, and the size of the tuyeres.

It will be seen that this pressure varies as the fourth power of the diameter of the tuyeres, so that it requires over twice the pressure to force the blast through 5-in. tuyeres under given conditions as to force it through 6-in. ones.

Other things being equal, the loss of pressure in passing the tuyeres varies as the square of the velocity, and the energy of the blast-jet also varies as the square of its velocity, hence the energy varies directly as the difference in pressure. This energy represents the ability of the blast-jet to overcome resistance in its line of action—in other words, its penetration; hence the penetration is proportional to the difference in pressure between the bustle-pipe or tuyere-stocks and the interior of the furnace. This explains the failure of many schemes of alternate small and large tuyeres, and tuyeres with flattened nozzles adapted to deliver a fan-shaped jet of air, which will cover the whole tuyere-plane equally (Gaines tuyeres). The difference of pressure, under given circumstances, within and without the tuyeres, is determined by the area of all the tuyeres, and hence the penetration in front of each of them is the same. Consequently, when alternate large and small nozzles are used, instead of the small ones throwing the air to the center and the large ones filling in the dead spaces between them, they all penetrate in figures of the same shape in the tuyere-plane; but the larger tuyeres make a larger and therefore longer one than the smaller —an effect the opposite of that intended.

Similarly the Gaines tuyeres distribute the blast so that the proper amounts are started in each direction to fill the area in that direction, but as some of these areas are much longer than others, while the difference in pressure is the same within and without the tuyeres in all directions, the penetration is either insufficient in some directions or too great in others.

For each diameter of hearth, number of tuyeres and general conditions as to blast-pressure, kind of cinder, etc., there is a given diameter of tuyeres which will throw the blast to the center sufficiently, without allowing great dead spaces on the walls between the tuyeres. If the tuyeres be made smaller than this, the force of the air-jet straight ahead will take an excess of blast to the center of the furnace and allow pilasters, so to speak, of dead material to form on the walls between the tuyeres, so reducing the active area of the hearth and making the furnace work too hot at the center. If, on the other hand, the size of the tuyeres be increased, the blast-jets fail to reach the center, but spread out laterally to such an extent that they may even cut the walls between the tuyeres, while a cone of dead material forms in the center.

One of the best furnace-men in the country once told me that he had banked his furnace, cut through the jacket, and put in additional tuyeres; that in cutting in for these they had found something like 3 ft. of perfectly dead material on the walls, and that after starting up they had made nearly 50 per cent more iron than the best they could do before. Naturally, he was a believer in multiple tuyeres, though not extreme in his views.

On the other hand many illustrations could be gathered in which the results were slight or even objectionable, under circumstances at first sight not very different.

The explanation is not difficult. In the first case the size of the tuyeres had probably been increased until they were as large as the blow-stocks and bustle-pipe connections could supply, and the velocity of the blast into the furnace could not be reduced by further increase in size of the tuyeres, although it was very much too high, with the blast all going up the center, and very heavy pilasters built up between the tuyeres on the bosh-walls, as shown by the depth of dead material cut through, before fire was reached.

When the increased number of tuyeres was put to work the velocity of the air-jets was diminished, and they were able to spread out laterally so as to melt off the pilasters, and increase the effective area of the hearth very materially. But if the bustle-pipe connections and tuyeres had been enlarged so that the proper quantity of air at the required temperature and pressure could have gone through them at the proper velocity, the results would have been in all probability almost if not quite as good as those attained by the use of a greater number of tuyeres.

If now, thinking to profit by this example, some furnace-man whose furnace was already equipped with a proper size of tuyeres, had followed a similar course, he would have so far reduced the velocity of the blast-jets that their penetration would have been insufficient; and they would have spread out sidewise, scouring the walls between the tuyeres and allowing a cone of cold material to form on the bottom in the center. In such a case, unless the area of the individual tuyeres had been reduced in proportion as their number was increased, the condemnation of multiple tuyeres would have been as hearty as its praise was in the first case.

It is not to be overlooked, however, that, other things being equal, the area of the pilasters on the walls varies nearly as the square of the distance between consecutive tuyeres or inversely as the square of the number of tuyeres. On the other hand, it must be remembered that the saving in hearth-area obtained by increasing the number of tuyeres increases much more slowly with each increase in their number, and that the difficulty and expense of maintenance of the tuyeres increases with each increase in their number. It is therefore not strange that the best opinion tends more and more strongly to a moderate number of tuyeres.

Before leaving this subject, I wish to call attention to a simple means of determining the activity of the different portions of the tuyere-plane, which is, of course, a direct measure of the penetration, and therefore of the correctness of the nozzle area. This method consists in passing a ¾-in. or ⅞-in. rod through one tuyere, and pushing it through the column of stock as quickly as possible, until it strikes the far side of the hearth, leaving it from twenty to fifty seconds, and quickly withdrawing it. The relative temperatures it has reached at different points will be a correct measure of the activity of the hearth at the corresponding points.

I am indebted to Mr. Austin Farrel of the Cleveland Cliffs Iron Company for this simple but effective expedient. When he told me of it three years ago, I believe it was very little known, though it has since come into use to some extent, but is probably still unknown to some who will be as glad to hear of it as I was.[1] The results shown by this test are sometimes surprising. I have seen a rod which had been put into a furnace, blown with about 11,000 cu. ft. of wind through six 6-in. tuyeres, come out almost dead black for 5 ft. in the center, and so nearly burnt in two at the noses of the tuyeres that it broke at those points when it hit the ground. (The tuyeres were 8 ft. from nose to nose.)

On the other hand, a reduction of the size of the tuyeres to 4 in., while it showed much better results on the rod and lowered the bottom of the furnace (which was high before), was succeeded after a few weeks by a considerably greater loss of tuyeres and coolers. This is believed to have been caused by the tops of the pilasters which formed on the bosh walls, making gutters, which led all the dripping iron and cinder down on top of the tuyeres and caused their more speedy destruction. The pilasters, of course, were formed by reason of the too great velocity and penetration of the blast jets, as before explained.

It seems very probable, therefore, that where furnaces have a small number of tuyeres, it is, on account of this secondary effect, unsafe to increase the penetration so much as to bring the test rod to a uniform temperature clear across the hearth. In furnaces with more tuyeres, the pilasters, in consequence of their smaller cross-section, do not extend so far up on the bosh walls, consequently this danger is much less serious. This is a point in favor of a larger number of tuyeres than six; but the necessity for this is now everywhere admitted, if large product be aimed at.

During the interval since this was written practice in regard to tuyeres has now settled down to a very moderate number of tuyeres for hearths even larger than were dreamed of at that time. Mr. H. A. Brassert has recently stated in a paper before the American Iron and Steel Institute, May, 1914, that furnaces with hearths as large as 16½ ft. or 17 ft. in diameter gave as good results with ten tuyeres as with fifteen, a statement which, in view of his experience, is authoritative.

The formula for the drop in pressure quoted above has been quite extensively used by some furnacemen to obtain comparative figures for penetration under different conditions. Mr. E. B. Cook of Cleveland in particular has made a careful study of this matter and has been good enough to communicate the results to me. He has found that the pressure-drop for small hearths ranges in good practice from 0.5 lb. to 0.75 lb., and on large furnaces from 0.8 lb. to 1.2 lb., except on the furnaces with very large hearths above mentioned, in which the

[1]Since writing the above, I have been informed by Mr. Frank Firmstone that this expedient has been described by French and German writers—in some cases, several decades ago.

figure is about 2 lb., a fact to which we shall refer later.

What is meant by penetration is not always easy to grasp at first sight. We think of the furnace as being filled with gas rather than as containing a violent current of gas, particularly in the hearth where the blast comes in horizontally and must quickly take a vertical direction. In order to make the state of things plainer we can conceive of a tube of glass, 6 in. in diameter, filled with broken coke, with say six jets of water coming into it near the top and flowing down through the coke under the influence of gravity.

Assume that the quantity of water per minute remains constant under all conditions and that first the nozzles of the jets are quite large so that the water flows into the tube with a very low velocity. We shall obviously have a condition such as I have tried to portray in Fig. 1, the current of water running principally down the sides of the tube and only reaching the center after it has travelled down a long way.

Imagine now that while the quantity of water remains the same, the jets are very much reduced in size so that their velocity is very great under the influence of a high pressure. We shall obviously have the condition shown in Fig. 2 in which the water is driven, in spite of the resistance of the coke, very largely to the center where the jets are stopped by impact upon one another. Here we shall have the greater portion of the water flowing down the center, and the sides not receiving their proportion for quite a distance down the tube.

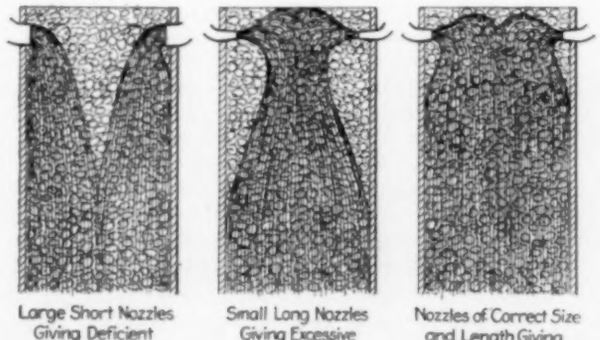
Obviously now we can establish an intermediate condition in which the velocity of the jets is considerable but not excessive, as shown in Fig. 3. They are given sufficient energy so that by the time they reach the center the energy is all exhausted by the resistance of the broken coke, and there is no surplus to pile up the water in the center and cause a disproportionate flow in that portion of the tube. At the same time the velocity is sufficient to permit only that quantity of the water to flow down the space next the walls which is proportionate to the area of that region. These are precisely similar to the conditions of deficient, excessive and correct penetration in blast-furnace practice.

The penetration test in the quotation above given, in which the rod was absolutely black all over except for a couple of feet near the ends where it was white hot was made on a furnace with tuyeres very large in relation to their number and to the amount of blast blown; this furnace formed a hugh "onion" of solid iron on the hearth which remained throughout the blast. The molten iron lay in the annular space between the onion and the crucible walls, and on account of the small area of this space stood at a considerable height therein at cast time; when the furnace was cast the iron flowed with the greatest violence for one or two minutes and was then followed by the cinder, the cast being over. When the furnace was blown out this hugh onion was found welded fast to the salamander, and rising some 2 ft. or more above the bottom of the hearth. There can be no possible doubt that this furnace worked continually with a cold center—a center column of stock which was not traversed by the gas, and which was only eaten away by a slow process of solution in the iron and cinder and stood like a cold poultice on the hearth. In subsequent blasts smaller tuyeres were used and the same conditions never arose again.

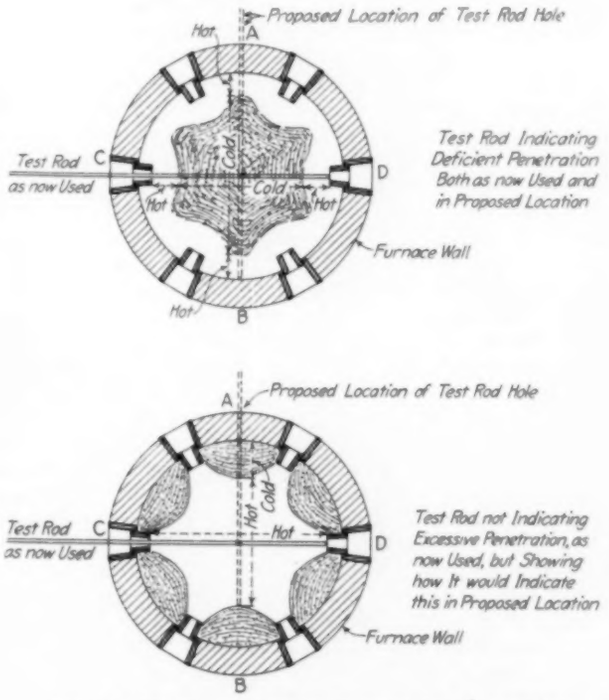
The operation of testing the penetration with an iron rod, described above as generally carried on, is rather unsatisfactory because no matter how excessive the supply of blast in the center may be it is obvious that there is no deficiency of blast supply near the walls in the space immediately in front of the tuyeres which the rod must traverse when the test is made through the tuyeres, the condition being somewhat as roughly indicated in Figs. 4 and 5, showing the condition in the tuyere zone of a furnace having six tuyeres with deficient and excessive penetration, respectively. Obviously the lines on which we should test are *AB* and not *CD*. This we can easily do, if there are cooling plates between the tuyeres and approximately on the level of the tuyere centers as there should be, by fitting the cooling plates half way between the tuyeres with a copper pipe, passing from the butt of the plate through into its nose, carefully screwed into the latter surface and pened over, the butt end of the cooling plate being caulked up against the copper pipe to make a tight joint.

This would give a water-cooled opening of sufficient size to admit a test rod and a penetration test made here would quickly show excessive penetration. These holes, of course, would be plugged with clay or could be closed with a screw plug, when not in use.

I have never tried this method of testing myself, but I see no reason to anticipate that it would not be per-


Large Short Nozzles Giving Deficient Penetration
Small Long Nozzles Giving Excessive Penetration
Nozzles of Correct Size and Length Giving Proper Penetration

FIGS. 1, 2 AND 3—CONSTANT QUANTITY OF WATER DISCHARGED THROUGH NOZZLES OF DIFFERENT SIZE INTO TUBES FILLED WITH BROKEN MATERIAL, ILLUSTRATING THE VARIATIONS IN BLAST PENETRATION



FIGS. 4 AND 5—LOCATION OF TEST ROD HOLES IN PENETRATION TESTS

fectly successful, and it should certainly give much more accurate information than testing through the tuyeres.

The Blast Pressure Required to Overcome the Resistance of the Charge Column

This is obviously a subject not open to accurate mathematical investigation of the kind that Fleigner was good enough to make in order to supply the means for determining the pressure drop through the tuyeres. We can never hope for such a solution since from the nature of the case the conditions are not only of infinite variety but change from instant to instant, and the pressure required to blow a given furnace is something in which no rule has ever been suggested for even approximate guidance. Nevertheless this also follows a natural law, and in spite of the variation in the conditions is amenable to mathematical expression, the constant, of course, being derived entirely from practice.

Let us consider first the relative velocities of the gas column at different zones of the furnace as affected by its volume and by the area of the furnace at the corresponding zone. The three zones at which we have any knowledge are the tuyeres plane: the top of the bosh, where we assume that combustion to CO is complete and that the gas has given up heat generated by its combustion down to the critical temperature; and the stock line of the furnace. The volume of the gas is determined by three circumstances, its chemical condition, its absolute temperature, and its absolute pressure.

Let us assume that the tuyere pressure is 15.3 lb. or 30 lb. absolute, that the top pressure is 0.3 lb. or 15 lb. absolute, and that the pressure at the top of the bosh has dropped in the same proportion as the fraction of the total height traversed, let us say one-fifth to 12 lb. or 27 lb. absolute. Let us assume further that the blast temperature is 1200 deg. Fahr., and the top temperature 400 deg. Fahr., the critical temperature prevailing at the top of the bosh, as before stated, being 2750 deg. the absolute temperatures corresponding to these are 1660 deg., 860 deg. and 3210 deg. Let us further assume in accordance with average practice that the top gas contains 60 per cent nitrogen.

At the tuyere plane the blast obviously enters as air; at the top of the bosh the oxygen of the air has become CO which increases the gas volume, as compared with air, from 1 to 1.207. At the stock line the nitrogen which is the same in absolute amount has fallen from 79.3 per cent to 60 per cent, so that the total volume must have increased by chemical action, additions, etc., to 1.32. Multiplying together the results of these three conditions we find that the relative volume at the tuyeres is $1 \times 1660/30 = 55.4$; the volume at the top of the bosh is $1.207 \times 3210/27 = 143$, and the volume at the stock line is $1.32 \times 860/15 = 75.5$.

If the gas is to travel at a uniform velocity the area at these levels must be proportional to its absolute volume at these points, and in a circular column the diameter of the column must be proportional to the square root of these volumes. These square roots are: 7.46, 11.95 and 8.67. Multiplying all of these by a constant factor to bring them to recognizable dimensions they become 16.1, 25.6 and 18.6. These are not very far from the diameters at the tuyeres, the bosh and the stock line of a 90-ft. furnace though the bosh diameter is too large and the hearth a little small.

We see, therefore, that if the furnace were empty the gas column would pass through it at an approximately constant velocity from top to bottom; the same would also be true if the percentage of voids if all portions of the furnace was the same. It is impossible for us to assert that this is true but I am unable to perceive any reason why we should not expect that it is approximately true since the variation around the top of the bosh is explained by changes in other conditions, as we shall presently see.

A heterogeneous mixture of materials of different sizes must, by mixing, reach a sort of irreducible minimum of voids depending on the size and kind of the materials. From the top of the bosh down to the tuyeres the coke is burned, and as the process is necessarily progressive the average sizes of the pieces at the tuyere zone must be much smaller than those at the top of the bosh which in a general way leads us to expect a proportional decrease of the voids, which would naturally necessitate a diameter of hearth larger than the gas volume alone would account for.

The materials other than coke which have existed as solids down to the top of the bosh are liquified as they pass below that zone with a considerable diminution of volume, and a much great diminution of their resistance to the passage of the blast on account of the closeness of their contact with the coke when in the liquid form, hence the available volume of voids is greater than in any other zone of the furnace either above or below and the area may be proportionally smaller than the gas column estimate would lead us to expect.

Considering these various conditions we may tentatively assume that the velocity of the gas column upward through the furnace is constant, though this is not necessary to the correctness of the formula for blast pressure, presently to be developed.

Turning now to the matter of the resistance to the passage of the gas offered by the stock column it will perhaps be as easy to approach it by the path I first followed as by any other.

A number of years ago there was an active controversy in one of the technical journals concerning formulas for the flow of compressed air in pipes. Numerous formulas were proposed and the results obtained by applying them to the same problem were truly startling. It finally reached the point where one correspondent declared that the initial pressure for a given flow in a long pipe line could be dropped from 160-lb. to 100-lb. gage pressure by the simple expedient of *raising* the terminal pressure of the air leaving the pipe line from atmospheric to 40-lb. pressure.

At this point I began to wonder if there might not be something wrong with the formula which all the parties were using in one shape or another, and with one set of constants or another. This formula was merely the mathematical expression of the statement that the friction of a fluid through a conduit is proportional to the length of the conduit, to the square of the velocity, to the wetted perimeter of the conduit, divided by its area, and to a coefficient depending on the liquid. This is a fundamental law and is absolutely correct, but a little consideration showed that when the air in the above example dropped from 180-lb. pressure absolute to 15 lb. absolute, its volume and therefore its velocity increased twelve times, and as the friction is by the law proportional to the square of the velocity the friction in this part of the pipe would obviously be 144 times as great per unit of length of pipe as in the high-pressure portion, except for the effect of the density.

To see what effect this consideration would have on the results I took one of the examples quoted and divided the length of pipe given into ten sections, figured the drop of the first section, then with the new velocity consequent on the increased volume due to the decreased pressure calculated the pressure drop for the second section, and so on, finally reaching a reasonable result. It presently became obvious that this was a legitimate field for a little calculus.

It is admitted that the friction of gases is proportional to their density, as it obviously should be, other things being the same, so we must introduce this factor into the fundamental law which may be expressed as a formula by putting

P_1 = the initial pressure absolute.
P_2 = the final pressure absolute.
L = length of the pipe.
D = the diameter of the pipe.
W = weight of air passing per minute.
S = density of air.
V = volume of free air per minute.
v = velocity in pipe per minute.

$$(\pi D) \Big/ \left(\frac{\pi}{4} D^2\right) = 4/D = \text{wetted perimeter} \div \text{area.}$$

K is a constant to be determined by experiments; all constant factors are thrown into this and may be disregarded until the final equation is reached, proportionalities being treated as equations.

Then

$$V = W/S$$

and

$$v = V \Big/ \left(\frac{\pi}{4} D^2\right) = W \Big/ \left(\frac{\pi}{4} D^2 S\right)$$

Then the fundamental law for a short section of pipe becomes

$$P_1 - P_2 = \frac{4kLSv^2}{D} = \frac{4kLS}{D} \times \frac{W^2}{(\pi/4)^2 D^4 S^2} = \frac{64kLW^2}{\pi^2 SD^5}$$

but S is proportional to P and we can put

$$P_1 - P_2 = \frac{64kLW^2}{\pi^2 D^5 \times (P_1 + P_2)/2}$$

For the very short length of pipe dL we have

$$P_1 - P_2 = dP \text{ and } (P_1 + P_2)/2 = P$$

so that we have

$$dP = \frac{64kW^2}{\pi^2 D^5 P}\, dL \text{ or } PdP = \frac{64kW^2}{\pi^2 D^5}\, dL$$

whence

$$P_1^2 - P_2^2 = K\frac{W^2 L}{D^5}$$

By applying the results of the extensive experiments on the flow of compressed air in pipes at the Mont Cenis tunnel I derived the necessary constant and the formula took the final form of

$$P_1^2 - P_2^2 = 0.0006\frac{W^2 L}{D^5}$$

in which form it has been very widely used, and extensive experiments made in the natural gas lines have completely verified its correctness when the proper factor for the density of natural gas is used.*

I had made a number of attempts to understand the relations of changes in blast volume to changes in pressure on a given furnace, and also the relation between the pressure required for a given volume of blast on two furnaces of different sizes, but I had never been able to obtain any results at all in accordance with the facts. Gradually it began to seem that the same laws might be applied in this case as in the case of the friction in pipes. That is to say, it was a case of gaseous friction with all the consequence of changes in velocity due to changes in pressure, but with this important difference that the friction in case of the round conduit comes only from the perimeter of the pipe, whereas in a furnace shaft filled uniformly with solid material the ratio of surface to volume of interstices is the same over the entire area, and therefore the factor for wetted perimeter is not needed.

Let us consider two furnaces of the same relative proportion, one, A, 50 ft. high by 12½ ft. bosh diameter, of 4000 cu. ft. capacity and blown with 8000 cu. ft. of wind per minute. This would require, we know by experience, about 3½ lb. pressure. The other, B, 100 ft. high by 25 ft. bosh diameter, of 32,000 cu. ft. capacity and blown with 64,000 cu. ft. of wind, what pressure will be required for it?

Let us assume that the two furnaces are each divided into fifty equal zones, those of A being similar to those of B, though only half as high. It is obviously reasonable to assume that the specific resistance of the stock to the passage of the gas will be the same in the corresponding zones; that is, that the ratio of interstices to surface of stock exposed will be the same in corresponding zones in each. Let us assume for the present that the shape of the furnaces is such that the velocity of the gas through them is constant in all zones, or if not, that it varies similarly in the two stacks so that this variation can be eliminated from consideration. This reduces the fluid friction of the gas current down to the same basis as that for flow in a uniform conduit.

Then proceeding as in the previous case we put

D = furnace diameter.
H = furnace height.
P_1 = absolute pressure at the tuyere zone.
P_2 = absolute pressure at the stock line.
W is the wind blown per minute measured under atmospheric conditions.
S = density of the air (proportion to the pressure when the temperature is the same).
W = velocity of gas through the furnace proportional to $\frac{W}{SD^2}$

Proceeding as before we find for a short section dH that

$$dP = K\frac{W^2 S}{S^2 D^4} \times dH = K \times \frac{W^2}{SD^4} \times dH$$

If we substitute P for S, to which it is proportional, we have

$$PdP = K\frac{W^2}{D^4} \times dH$$

and by integration

$$P_1^2 - P_2^2 = K\frac{W^2}{D^4} H$$

The constant K we determine from the pressure assumed from general experience in the case of the small furnace, and with this value of K we determine the pressure in the large furnace to be 19.4 pounds as follows:

For our small furnace, $P_1 = 18.5$, $P_2 = 15$, $W = 8000$, $H = 50$, and $D = 12\frac{1}{2}$, from which we can find K as follows:

$$(342 - 225) = K\frac{64{,}000{,}000 \times 50}{24{,}400} \text{ or } K = 0.00089$$

and for the large furnace using the same value of K we get

$$P_1^2 - 225 = 0.00089 \times \frac{64{,}000^2 \times 100}{390{,}625} = 1{,}050{,}000 \times 0.00089 = 935$$

$P_1^2 = 225 + 935 = 1160$, $P_1 = 34.1$ absolute $= 19.4$ gage.

It must be thoroughly understood that the value of K is derived entirely from experience. There is no known method by which we can even attempt to determine its value mathematically. We must therefore determine a value for it which will fit established practice reasonably well before we can safely use the formula or com-

*I subsequently found that this formula had been developed at least once and probably several times before, but apparently the earlier investigators did not take the trouble to determine the constant, without which the formula is practically valueless, and it had never come into general use.

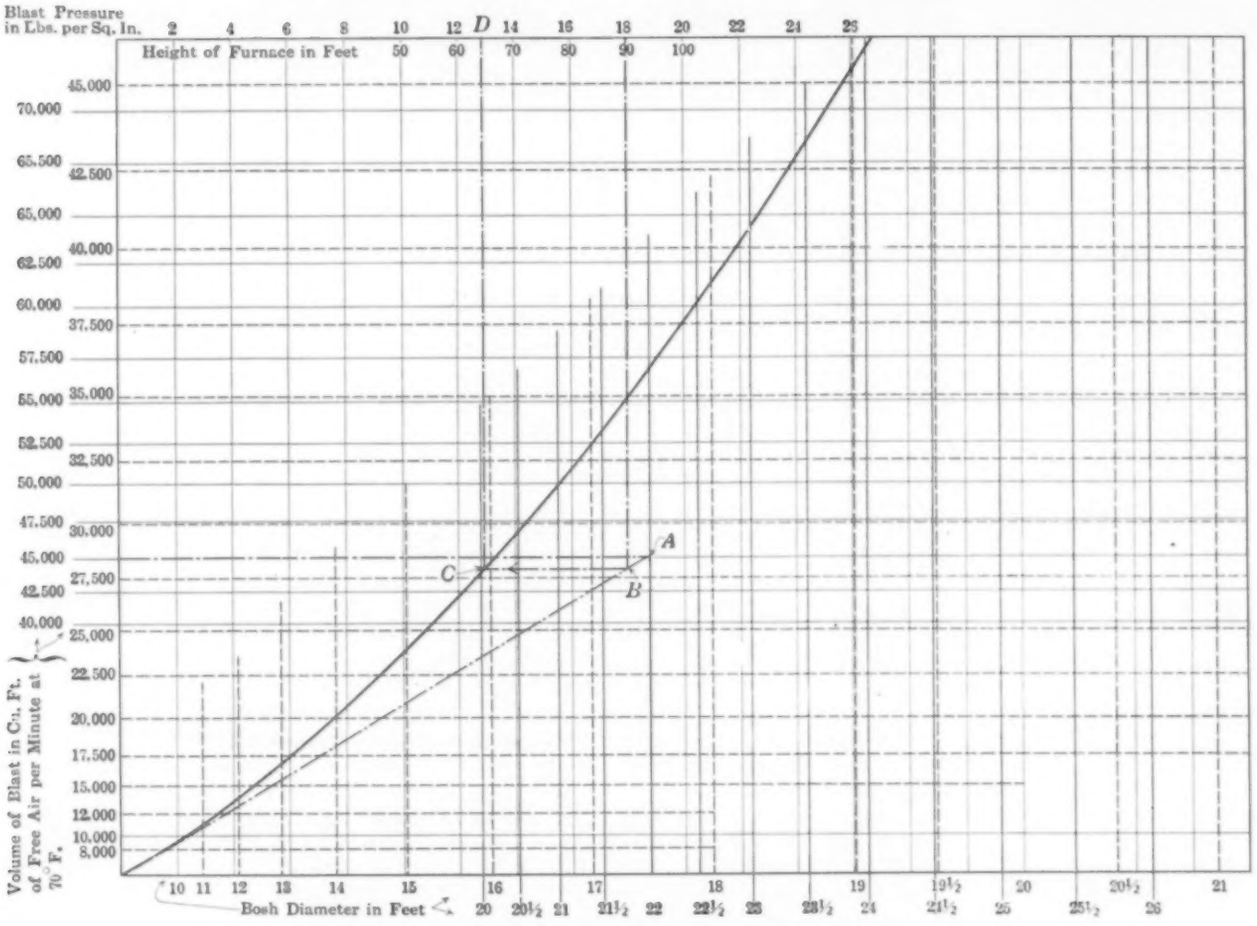

Blast Pressure in Lbs. per Sq. In.
Height of Furnace in Feet
Volume of Blast in Cu. Ft. of Free Air per Minute at 70° F.
Bosh Diameter in Feet
A
B
C
D

FIG. 6—BLAST-PRESSURE CHART

Instructions for Use of Blast Pressure Chart:—Note first of all that there are two rows of figures along the bottom giving bosh diameters, and two rows along the left-hand end giving blast volumes. The inside rows of these must always be used together, or the outside rows; never figure from the inside in one row and from the outside in the other row. Some of the figures are duplicated in the inside and outside rows. This is to extend the scale and flexibility of the chart. As long as the corresponding rows are used together it makes no difference which one is taken.

Take the vertical through the figure at the bottom giving the bosh diameter and the horizontal through the figure at the left giving the blast volume in cubic feet of piston displacement. From the intersection of these draw a diagonal through the origin and extend it until it cuts the vertical through the height of the furnace given just inside the cross-sectioned area of the diagram at the top. From this intersection draw a horizontal to the curve, and from the point of intersection draw a vertical to the top of the diagram and read the pressure in the figures along the top edge of the cross-sectioned area. For instance, find the normal working pressure of a furnace, 22 ft. bosh diameter by 90 ft. high, blown with 45,000 cu. ft. of wind piston displacement per minute. The vertical through 22 in the outside row intersects the horizontal through 45,000 in the outside row at the point A. Through this draw a diagonal toward the origin until it cuts the vertical through 90 ft. furnace height at B. From this draw a horizontal to its intersection with the curve at C, and from this draw a vertical to the point D at the top of the diagram, which shows a little less than 13 lb. pressure.

pare the pressure of different furnaces with its aid. By comparisons with actual results I reached the conclusion that the proper value of K was 0.00070, and in order to make the formula easy of application I prepared a diagram, shown at Fig. 6, with the aid of which, with the accompanying explanation for its use, problems involving furnace pressure may be solved in one minute.

Through the kindness of a number of furnacemen this formula has been checked against practice on a number of furnaces, with the result that I believe its indications are about a pound too low in the region of 15 lb. pressure, and it is probable that the value of K should be 0.0008. While there are variations, the correspondence between the results of this formula and the pressures actually found at furnaces of different sizes and operating under different conditions is very much closer than might be expected, so that I believe it may be safely assumed to be founded on correct hypotheses, and in the absence of knowledge based on practice to give a reasonably reliable indication of the actual pressure that may be expected under any given conditions.

It will be noted that this pressure includes both the stock column resistance and the tuyere pressure. It would be possible to subdivide it so as to give stock column resistance only, but this is not worth while for two reasons: First, the variations of tuyere pressure drop are smaller than the normal variation in blast pressure. Second, the tuyere pressure drop increases with the size of the furnace, as stated above, so that its variation is roughly the same as that of the stock column pressure. It is therefore not worth while to differentiate them, but is simpler and is exact enough for all practical purposes to include both in the one formula.

The top pressure assumed for the chart is that of the normal atmosphere, 14.7 lb., on the basis that most of the furnaces in the country are located several hundred feet above tide level, and the actual atmospheric pressure is about enough lower than that at sea level to allow for the top pressure required to drive gas through the flues.

It should be noted that variations from this formula tend to be, to a certain extent, self-compensating. That is to say, if owing to any condition the pressure rises, this in turn increases the density, which reduces the velocity and thereby tends to keep down the pressure drop. This may be illustrated by the electrical resistance of the tungsten lamp as compared with the carbon. The resistance of carbon decreases with temperature, therefore a slight rise in voltage, driving more current

through the filament, heats it hotter, reduces its resistance and tends to let still more current pass, whereas with the tungsten lamp the resistance increases with temperature, so that an increase in voltage results in an increase in temperature, and this in turn in an increase in resistance, thereby tending to cut down the increased flow of current and reduce the variation due to its effect. As a result tungsten lamps show much smaller variations of luminosity with a given variation in voltage than do carbon lamps under the same conditions. So owing to the nature of gaseous friction variations in blast pressure are much smaller than they might be expected to be.

This fact probably has something to do with the comparative correctness of the blast-pressure chart over wide variations in conditions.

It should be clearly understood that the pressure given by this formula is only the normal working pressure. The furnace may get into a condition in which the gas blows through easily, due, perhaps, to a scaffold on one side above and another on the opposite side below, or something of that kind, and in such comparatively rare cases the actual pressure will be lower than that shown by the chart.

On the other hand, very many causes may make the furnace work with a considerably higher pressure than that given by the formula; there is, in fact, no limit in this direction except the power of the engines. First, the furnace may form a ring scaffold which forces all the gas to go through a sort of nozzle much smaller than the diameter of the furnace, and consequently with a much increased velocity and resistance. Second, the slag may become tough and stringy, and therefore be blown by the blast into the interstices of the coke, greatly obstructing them instead of running in quiet trickles down over the surface. Third, the hearth and bosh in case of serious trouble may become almost frozen solid, with the result that only very small passageways are left through which the blast can find its way, and the slag in those is excessively tough and stringy. Fourth, and probably most common of all, the deposition of carbon (described in the article on chemical principles), which is sometimes extremely voluminous, in cases amounting to several times the bulk of the ore, may be deposited in the interstices of the stock in such quantity as to greatly obstruct the flow of the gas. These are all conditions which we have difficulty in proving; to measure them quantitatively is, of course, inconceivable. We cannot therefore hope to fit a formula to these diverse variations.

Leaving abnormal conditions to one side it has been found at a number of plants that the formula coincides with the results of normal-working furnaces with a degree of accuracy which is fairly satisfactory in so approximate a matter, though, as before stated, it is perhaps a little low. It should be stated that the quantity of air given on the chart is that measured by piston displacement in normal practice. This is indefinite to a certain degree, but unfortunately we have nothing else, and with the great improvements of blowing engines which have occurred in recent years this is sufficiently accurate for our purpose. It is probable that the actual air corresponding is less by about 12 per cent under the ordinary conditions of good practice.

It will be noted that this formula takes no account of difference in the condition of the ore, which varies from the fine sand of the Mesabi to the all-lump ore of Southern practice. There is probably a slight variation due to this cause, but owing to the self-compensating nature of the action described above, it is very much less important than might be expected, or at least the formula fits furnaces under Southern conditions about as well as it does those running on Lake ore, which means generally from 50 to 90 per cent Mesabi.

It is greatly to be hoped that all interested will apply this formula to their conditions, and if they will communicate the results of such application to the editor of this journal, with as full a description as possible of the conditions, it will promote the collection of data which will either put this formula on a firm foundation or upset it altogether. It is presented here because of the absolute lack of any other means of reaching the desired end, and because the results of its application were found to coincide so much more broadly with practice than was expected when the formula was developed.

The Effects of Blast Pressure

The pressure of the blast tends to retard and make more difficult the descent of the stock. It is curious that this idea is not more generally held (a few years ago it was almost entirely ignored), but on the basis of Newton's first law of motion, that "action and reaction are always equal and in opposite directions," it seems impossible to escape the conclusion that if the stock column resists the ascent of the blast at the rate of so many pounds per square inch, then necessarily the ascent of the blast resists the descent of the stock by an equal number of pounds per square inch.

This view has not always been admitted from the theoretical side, but practically it has been recognized by the century-old practice of slacking the blast when the charge column was "hung" and refused to settle properly.

From this consideration it is very evident that to secure free and regular settling of the charge one of the prime conditions is to have the blast pressure low in relation to the weight of the charge column, and this, with the great expense involved in compressing the blast to a higher pressure than is absolutely necessary, would lead us to believe that high pressure was an unmitigated evil, and that in general the lower the pressure with which a furnace will work, the more satisfactory its work will be. But while this is a fundamental consideration, and should be the controlling one, there is another condition brought about by high pressure which is worthy of careful consideration, as first pointed out to me by George G. Crawford. This consideration is that the velocity of combustion is greatly augmented by pressure, and as most modern furnaces are driven to the limit set by their ability to burn the carbon in the space available for that operation this is a consideration not lightly to be neglected. Moreover, quick combustion means a decided localization of the zone of highest temperature, which in turn means lowering the zone of fusion and bringing more closely under control that region of the furnace at which scaffolding due to pastiness is likely to arise.

The standardization of bomb calorimeters is the title of *Circular* No. 11, issued by the Bureau of Standards. The paper describes briefly the methods of calibrating and using bomb calorimeters. Provision is made by the Bureau for distributing standard samples of pure materials, such as sugar, naphthalene and benzoic acid, for the purpose of standardizing calorimeters and securing uniformity in heat tests.

The Haynes Stellite Company of Kokomo, Ind., has placed a contract with the Snyder Electric Furnace Company, Chicago, for a 1½-ton per 24-hr. electric melting furnace which has a one-eighth-ton holding capacity, 50 kw. input and will produce 12 heats per 24 hr. The Haynes Stellite Company is the manufacturer of the cutting alloy "stellite."

The Safe Transportation of Explosives and Other Dangerous Articles*

BY COL. JAMES L. TAYLOR

Before the organization of the Bureau of Explosives, as it is commonly called, this important matter was regulated through the local regulations of the railways, and, as might have been expected, such regulation was often found to fail of its purpose in the particulars of that standardization and uniformity so indispensable to the enforcement of regulations which instituted or attempted to institute departures from the prevailing methods of competition and of practices that had previously been permitted to exist. The consequences of these conditions were in too many instances disastrous, and the progress toward their amelioration was of a very slow and disappointing character until the organization of the Bureau of Explosives was effected and its regulations put into force by practically all of the railways embraced in the organization of the American Railway Association.

Following this condition as a natural sequence, it was finally adjudged necessary to have the matter placed in the category of federal control, and by appropriate legislation to make the enforcement of the regulations one of universal and standard requirement. And so on May 30, 1908, the President of the United States approved an act of Congress under the title, "An act to promote the safe transportation of explosives and other dangerous articles and to provide penalties for its violation." The Interstate Commerce Commission was made that branch of the government to whom was entrusted the fixing and promulgation of the regulations to make the act effective within ninety days after its passage. Well within the limit of that period the regulations were duly promulgated, which in the language of the act were to be "the best known practicable regulation" for the purposes of the act, and which, in fact, were in a large degree the regulations of the American Railway Association, with some modifications which time and experience had demonstrated as necessary. They were issued in the name of and by the authority of the commission as the regulations to govern the transportation of the articles referred to.

On March 4, 1909, the foregoing act of Congress was repealed and the date of such repeal was to be on Jan. 1, 1910, and the title of the repealing act is a very suggestive one, for it is entitled "An act to codify, revise and amend the penal laws of the United States, to take effect and to be in force on and after Jan. 1, 1910." From that date to the present the legislative regulation of this matter has been one of federal law. The sections of this act last named, which apply more especially to the subject of our consideration at this time, are included in Sections 232 to 236 inclusive. As the first three are practically the first act in its essence, I will not repeat them. Sections 235 and 236, however, I will repeat, as they stand upon the record of the law books to-day, and have stood there since Jan. 1, 1910:

Section 235. Every package containing explosives or other dangerous articles when presented to a common carrier for shipment shall have plainly marked on the outside thereof the contents thereof; and it shall be unlawful for any person to deliver, or cause to be delivered, to any common carrier engaged in interstate or foreign commerce by land or water, for interstate or foreign transportation, or to carry upon any vessel or vehicle engaged in interstate or foreign transportation, any explosive, or other dangerous article, under any false or deceptive marking, description, invoice, shipping order, or other declaration, or without informing the agent of such carrier of the true character thereof, at or before the time such delivery or carriage is made. Whoever shall knowingly violate, or cause to be violated, any provision of this section, or of the three sections last preceding, or any regulations made by the Interstate Commerce Commission in pursuance thereof, shall be fined not more than $2,000, or imprisoned not more than eighteen months, or both.

Section 236. When the death or bodily injury of any person is caused by the explosion of any article named in the four sections last preceding, while the same is being placed upon any vessel or vehicle to be transported in violation thereof, or while the same is being so transported, or while the same is being removed from such vessel or vehicle, the person knowingly placing, or aiding or permitting the placing, of such articles upon any such vessels or vehicle, to be so transported, shall be imprisoned not more than ten years.

These are the requirements of the law which must be complied with, and a full understanding of the law is requisite which must be had by every person in any way brought into connection with the transportation of these dangerous articles.

The "other dangerous articles" (besides explosives) are chiefly inflammable gases, inflammable solids or oxidizing materials, acids or corrosive liquids, compressed non-inflammable gases, etc.

Probably the most important feature is the safeguarding of these dangerous articles by the security of their packing. And I am happy to state that in this respect the Bureau for the Safe Transportation of Explosives and Other Dangerous Articles has achieved a notable success in securing, after a somewhat prolonged effort, the full support of the manufacturers and shippers of acids, charcoal, friction matches, compressed gases and other articles of chemical industry, most of whom have co-operated with us by giving us the full benefit of their experience, especially in the matter of formulating suitable rules for transportation.

In a similar way, too, we have secured in a very large degree the co-operation of the manufacturers of shipping containers for these classes of dangerous articles, especially in railway transportation.

It is well known that there is no well-defined limit to the forces that may act upon the shipping containers which include the crushing weight of other packages piled upon them; the shifting of freight in the cars incident to transportation conditions; the failure of many types of boxes of various kinds to carry the load of other packages; the proper resistance to the shocks incident to the handling of cars, and the proper methods to resist the force of such shocks, by a proper cushioning of containers to resist them, as well as the high pressures slowly applied.

In this connection I would refer to a series of monographs upon this subject: The lectures of Col. B. W. Dunn, late of the Ordnance Department of the United States Army, who is the chief inspector of the Bureau of Explosives of the American Railway Association, delivered upon many occasions, among which were the lectures at the twenty-fifth semi-annual convention of the National Association of Box Manufacturers, at Detroit, Aug. 28-30, 1912; the address at the first annual convention of Manufacturers of Corrugated Fiber Containers, held in Atlantic City, July 7-9, 1914, and finally, and perhaps the most important of all in the profitable character of the information which it gave, the address of Colonel Dunn which appears in the Proceedings of the New York Railroad Club at its meeting April 15, 1910, under the title "Stresses Developed by Collisions of Freight Cars."

*Synopsis of a lecture delivered before the First National Exposition of Chemical Industries, New York City, on Sept. 23, 1915. The author is assistant to the chief inspector of the Bureau of Explosives of the American Railway Association. The address was illustrated by numerous stereopticon views.

Abrasive Materials.—The production of abrasive materials in 1914 is given in Bulletin II:29 of the Geological Survey.

The Barium Industry in the United States Since the European War*

BY MAXIMILIAN TOCH

Within the last eighteen years there have been three attempts to establish a barium industry in the United States prior to Aug. 1, 1914, and all three of them have been unfortunate failures. I have investigated the cause of these failures and I find that two of them failed through financial and chemical mismanagement, while the third was utterly destroyed on account of its inability to compete against foreign competition.

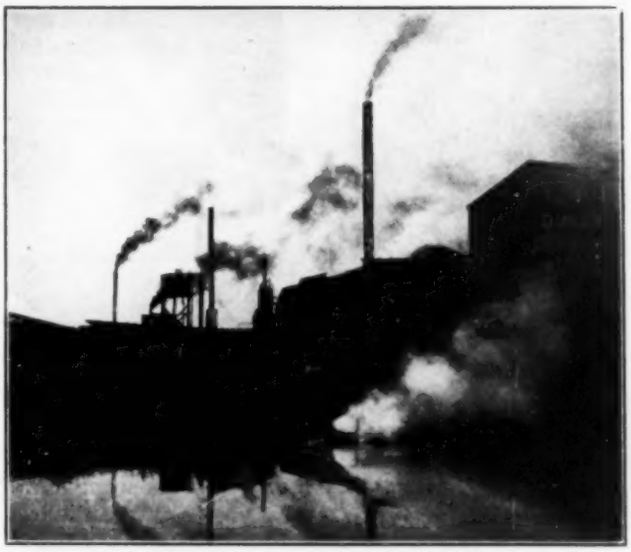
I have been identified with the manufacture of barium products for nineteen years, and while the venture of my firm has been successful it was never very large and was confined to one particular barium salt, which sold entirely on account of its quality.

A well-known and successful chemical corporation manufactured barium chloride and barium sulphate in New York City a little over fifteen years and made very good products. As soon as they turned out reasonable quantities some foreign chemical manufacturers undersold them at such a price that they found they were unable to continue, and as soon as they were put out of business the foreigners raised their price. They have kept the trade ever since, for they have taught the American manufacturer the unfortunate lesson that if he were to establish an industry he could not continue it successfully, for the well-known methods of "dumping" large quantities on this market in addition to the meager tariff would soon teach them that they had better not try to make American goods again.

I believe I was the first one who pointed out to the Department of Commerce the shortcoming of our own laws. If a manufacturer in this country attempts to put a competitor out of business by what is known as unfair methods, all one has to do is to file a complaint with the Federal Trade Commission, and the matter is considered at once by its examiners and attorneys. But it is quite obvious that any law made in this country has no sanction against a foreign corporation outside of the jurisdiction of the United States any more than a firm in the United States can be held responsible for its actions according to the law of England, France or Germany.

Two weeks after the great war in Europe was under full swing, my firm recognized that a very interesting portion of its business would be obliterated if we could not obtain barium compounds, and, to be exact, on the 14th of August, I started negotiations for a plant at Sweetwater, Tenn., for the purchase of a large tract of land containing the mineral barite, and in October, just a year ago, the first carload of barium salts was shipped from Sweetwater, Tenn., to our Long Island City works. From that day to this the plant has been working night and day. Additional mines have been purchased, additional installation has been erected, and the corporation is destined to continue to grow provided our own parsimonious and short-sighted Government will not aid in its final destruction.

*A paper read before the New York Section of the American Chemical Society, Chemists Club, October 8, 1915.



PLANT OF DUREX CHEMICAL WORKS AT SWEETWATER, TENN.

There has been a great deal of talk about the dye industry in the United States, and any number of capitalists would invest the necessary funds to-morrow to establish a permanent aniline industry in the United States if they had some kind of assurance that the laws which prohibit unfair competition among ourselves could in some manner be made active as against those outside of the United States. There has been a tremendous amount of talk with reference to enacting an "anti-dumping" law. For months this has gone on, and up to now it is simply nothing but talk. A great deal of publicity has been given by public officials about the new and wonderful methods and short cuts that have been discovered for the manufacture of aniline dyes out of soft coal to such an extent that it has made the finest and best method for stock jobbing that has ever existed, but there isn't a color maker in the United States who can buy for immediate delivery the necessary aniline colors for the manufacture of the lake colors in any reasonable quantity in spite of all this caloric atmospheric discussion.

The barium industry in the United States is not an industry that can be started over night, and among the failures that I referred to, one of them was due to the fact that theory took the place of practice. The starting point of the barium industry in the United States is the admixture of barium sulphate and coal, both of which are finely divided, and heated, when the coal converts the entire oxygen into carbon dioxide, and leaves barium sulphide behind. The best that any man can do in practice is to get 70 to 75 per cent reaction. In our works we have had runs where we have gone as high as 85, but I think this is purely accidental, yet theoretically the reaction ought to be complete; and the difference between theoretical chemistry and chemical engineering shows that the chemical engineer must devise apparatus which will fit his raw material.

PILE OF ORE OUTSIDE FACTORY

Not all barite is suitable for reduction. I have seen disastrous results where calcium fluoride was contained in the barite, and where certain forms of iron would actually produce a paralysis in the reaction, and if up to now I have had a slight measure of success in the manufacture of these compounds it is because I had a knowledge of the manufacture of these compounds prior to the European war.

Chemical engineering is 49 per cent chemistry and 51 per cent business acumen. I place the business end ahead of the chemical end for the obvious reason that ability to finance, ability to sell at a profit, and ability to organize are ahead of the chemical reactions involved. You can buy chemical talent, but you cannot buy business efficiency. You can buy barytes of any kind, but you cannot buy in the open market the skill necessary to design this apparatus so that the results will be uniform. I have seen a chemical industry fail because no commercial method of water purification necessary for the reaction was at hand, and if the subject had been carefully studied in advance a fortune could have been saved.

There are many arguments in favor of protection of the chemical industries in the United States, and what I say about the barium industry might refer practically to any other chemical industry, for we have economic differences between Europe and America which cannot be neutralized excepting by a system of protection in addition to an "anti-dumping" law. The freight rate from Tennessee to the coast on our raw material is about $4.50 per ton. That is to say, barite cannot be shipped for less than that except by rail and water, which is a purely theoretical rate, for under existing circumstances a fifty-ton car of barium ore cannot be transferred to a steamer unless the material is packed in sacks or barrels. The same material pays $1.80 freight from Bremen to New York, and I have it on very good authority that the rate from the mines of Germany to New York is $2 per ton, because the German Government has always fostered its commerce by making special freight rates to export shippers.

The railroads in this country, which are, of course, all privately owned, show that they cannot carry trade for less than a fixed sum owing to their overhead charges; but their overhead charges, which start with the interest on their watered stock, are so great that the shipper must suffer. An eight-hundred-mile haul from Tennessee to New York pays $5 per ton, and a twelve-hundred-mile haul from Missouri to New York pays $4 per ton, and the one railroad can afford to do it and the other railroad cannot afford to do it. If the unsuspecting public is forced to support an overburdened railway, the Government should at least equalize this matter by giving us a protection in the form of tariff.

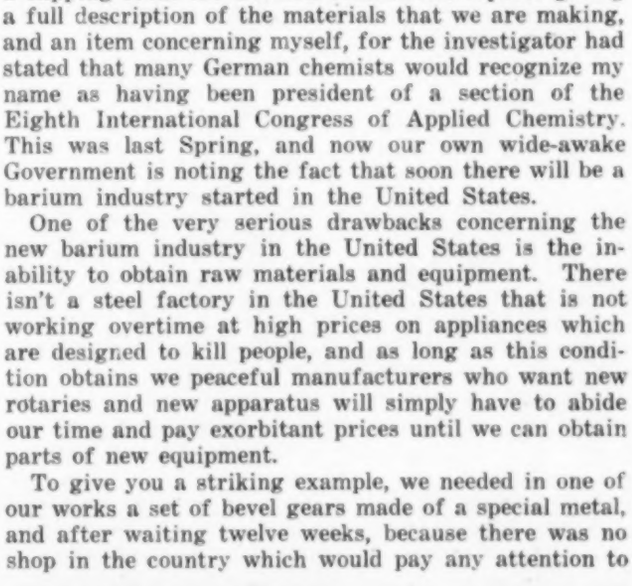
It is very amusing to read in the papers almost daily about wonderful new methods and new corporations who are about to manufacture all the dyestuffs in the United States and that the shortage of dyes and chemicals necessary for dyeing is about to disappear. It seems to be a pity that any Government employee will lend himself to the exploitation of what looks like stock jobbing schemes without due investigation, for I have not been able to buy any American dyes, and no one else that I know of has been able to buy any American dyes such as are generally used for the manufacture of lake colors. On the same sheet of the daily press in which these forthcoming industries are heralded you could invariably find a news item stating that the dye works are practically shut down for want of even simple dyes.

It is at once amusing and pitiable to note the superficiality with which these investigations have been conducted. Some weeks ago in the New York *Times* there appeared a statement that a corporation was formed and very shortly there would be no dearth of the barium chemicals necessary for the manufacture of mordants and paper glazing material, and that a great deal of credit should be given to this concern as being the first concern to manufacture the barium chemicals in the United States, and this just exactly one year after the industry with which I am connected had been steadily turning out hundreds of carloads.

It may be true that I have not trumpeted from the housetops that the concern which we started was supplying material with great regularity and of chemical uniformity, and our own Government did not know it. On the other hand, last Spring a friend of mine sent me a clipping from the German commercial reports giving a full description of the materials that we are making, and an item concerning myself, for the investigator had stated that many German chemists would recognize my name as having been president of a section of the Eighth International Congress of Applied Chemistry. This was last Spring, and now our own wide-awake Government is noting the fact that soon there will be a barium industry started in the United States.

One of the very serious drawbacks concerning the new barium industry in the United States is the inability to obtain raw materials and equipment. There isn't a steel factory in the United States that is not working overtime at high prices on appliances which are designed to kill people, and as long as this condition obtains we peaceful manufacturers who want new rotaries and new apparatus will simply have to abide our time and pay exorbitant prices until we can obtain parts of new equipment.

To give you a striking example, we needed in one of our works a set of bevel gears made of a special metal, and after waiting twelve weeks, because there was no shop in the country which would pay any attention to



TYPICAL BARITE MINE SHOWING CHUNKS OF PROJECTING BARITE ORE

us, we finally had to devise gears ourselves as a makeshift. Nor does this apply to machinery alone, for no one could take any contract on any of the barium salts like the chloride, nitrate or sulphate and hope to deliver any part, because it is absolutely out of the question to obtain acids. The people with whom we have dealt for our acids for years are simply either refusing to give us any materials or supplying us with ridiculously small lots at enormous prices, for there are other channels which buy larger quantities and pay more than the peaceful manufacturer can afford.

I mentioned before that industrial chemistry was 51 per cent business and 49 per cent chemistry, and these figures will serve for an example. I was often asked why we located at Sweetwater, Tenn., and while I did not have very much time to look around I found that that was the logical and strategic location for coal, labor, water, and ore, for all of these materials combined are perhaps more reasonable at that locality than anywhere else I know of. In order to make a success of this enterprise the works should be located within such a radius of the mine that the ore can be delivered at a reasonable rate.

The great and only question with reference to the barium industry is whether this industry shall live after the European war is over, and I could not answer that categorically without some explanation. It depends entirely upon our own Government. If on account of the economic conditions which I have described, foreigners will be prevented from underselling us, we will be able to live.

Our own manufacturers should get together and understand that the underlying principle of successful business is to have a profitable business, for no argument is necessary on this point other than to prove that when you have a growing and going manufacturing concern, in order to have peace and success, you must be able to pay your men a living wage, and you must be able to pay them regularly, for the shutdown of any works in any community, particularly in a small one, produces disastrous results. Innocent women and children are made to suffer through it, so that in the greed of obtaining business manufacturers must understand that cut-throat methods lead to bad results for all parties concerned. I hope that when this war is over, the barium industry will be firmly established and protected, as the economic conditions in this country would demand.

What a remarkable difference there is between the enterprise in Japan as compared to the United States. Japan has suffered with a lack of chemicals and dyestuffs just as much as we have, and both Houses of the Japanese Government have voted to subsidize all corporations and firms who are about to engage in the manufacture of chemicals and dyestuffs that were not formerly made in Japan to the extent of three and one-half million dollars, the other half to be subscribed by the manufacturers themselves. It is understood that the subsidy granted is for ten years in order that the new manufacturing concerns may pay a dividend of at least 8 per cent. In addition to this it is understood that the Japanese Government has protected these manufacturers in the form of a high tariff. And that is the country to whom we have been sending missionaries in order to convert them from their archaic beliefs.

It seems to me that it is a great pity that nothing has been done up to now to protect the American manufacturer, for a subsidy from the Government is out of all reason, even though this is at present the richest Government in the world and Japan rated as one of the poorest. As I said before, we can only hope to succeed if our Government will help us from being destroyed by outside influences.

Theories of the Flotation Process

In METALLURGICAL AND CHEMICAL ENGINEERING, Sept. 1, 1915, page 571, we presented a complete description of the Callow pneumatic flotation process, giving flow sheet and several illustrations of actual operations. Since that time Mr. J. M. CALLOW has presented further operating data and certain theoretical considerations, embodied in a paper read by him at a meeting of the Utah section of the A. I. M. E., Salt Lake City, from which we excerpt the following:

Theories

So far no satisfactory explanation of flotation phenomena has been advanced. At my instigation and under my direction, a large amount of research work has been done in an endeavor to formulate some logical explanation of the phenomena, and perhaps to find some scientific way of conducting flotation experiments in place of the empirical methods now in vogue. Although the latter object has not yet been attained, still these experiments have resulted in the formulation of a theory that appears to be well grounded and that may prove of interest and value to others engaged in this art.

Much work has been done at the Mellon Institute at Pittsburgh under the direction of Dr. Raymond C. Bacon, and lately by James A. Block at the station of the U. S. Bureau of Mines in Salt Lake City. The result of all of this work is summed up in the following statement:

In considering the connection between flotation phenomena and the physical properties of the minerals concerned, there are two parallelisms:

1. It has been noticed for some time that the minerals which floated were not easily wetted by water, while those which were easily wetted did not tend to come up with the froth. This is the basis of about the only theory so far widely circulated; it was well stated by Hoover in his book, *Concentrating Ores by Flotation.*

2. There is a parallelism between certain electrostatic characteristics and the flotation properties of ores.

In the first-mentioned theory, surface tensions and contact angles should be considered. Certain minerals, such as galena, will float on the surface of still water, while gangue particles, since they possess a greater adhesive attraction for the water than the water's cohesive attraction for itself, will be drawn through the surface film into the interior, and sink because of their greater specific gravity. These properties of flotable minerals and gangues are increased by the presence of oil and acid. Oil sticks to galena with greater tenacity than it does to silica, and an oil surface is still less easily wetted than a galena surface. The acid in the water causes a still greater difference in the various surface tensions. This, it seems, is without question the explanation of the action in the MacQuisten process, in which the ore particles are lifted to the surface, where they can be removed by skimming off the surface layer of the liquid.

With reference to the second parallelism, it has been noticed that extremely small amounts of certain colloidal impurities, such as saponin or tannin, were detrimental to flotation; while others, such as Congo red and methylene blue, did not interfere, and were, if anything, beneficial. In classifying these, the injurious ones generally came under the head of what physical chemists call electronegative colloids, while electropositive colloids were not harmful. Suspended particles will generally migrate when placed in an electric field, and this classification comes naturally from the direction of their migration. This migration is called electrophoresis, or electrical endosmose, and is the result of the formation of contact layers around the particles by the liquid

containing them, very similar to the formation of surface films when liquids come in contact with air. These contact films almost invariably have a difference of potential between their inner and outer surfaces. An air-water contact film has, for instance, a difference of 0.055 volts, and other contact films have similar charges. This causes the particles to act like charged solids, and to be attracted by electric charges of opposite sign.

The charges on solids and non-miscible liquids can be conveniently studied on the stage of a microscope.

This work naturally led to the study of the charges exhibited by various ores and minerals, and in that work an interesting parallelism was observed, namely, that flotable minerals seemed to have positive charges, and non-flotable gangues negative charges. Some gangues were found with positive charges, but they were characteristically hard to handle, having a tendency to come up with the froth. These charges sometimes vary with the acidity or alkalinity of the liquid, and this variation is not inconsistent with the effects of acidity or alkalinity on the flotation of ores.

It has been noticed that these electrostatic properties depend on the condition of the surface of the particle and not upon the composition of the mass. For instance, lead oxide which is ordinarily negative, or neutral, when covered with a sulphide coating, takes upon itself a positive charge.

Recent investigations on the coagulation and deflocculation of s[illegible]mes, on the coagulation and dispersion of colloids, and along similar lines, show that these contact film charges, although small, have an important bearing on the dispersion or coherence of particles suspended in liquid mediums. In fine suspensions and in colloidal solutions, these charges may often be neutralized by the introduction of oppositely charged ions, precipitation generally taking place whenever these charges fall below certain limits. Oppositely charged contact films have a general tendency to coalesce, while similarly charged films, if their charges are great enough to overcome natural cohesiveness, do not seem to coalesce, but to repel each other, and if the weight of the particles is small enough in relation to their size and surface, permanent dispersion will take place, the particles distributing themselves through a liquid in much the same manner that gas will fill a container.

In view of the above observations, it seems possible that flotation is the result of difference in polarity in the charges on the various ore particles, and on the bubbles. Since oil contact films and air contact films have both been proved to have negative charges, the positively charged minerals will adhere to either. The bubble mantles in a flotation machine are undoubtedly composed of oil, or of oil in emulsion, since pure water alone will not froth. The same forces, then, that cause oppositely charged colloids to agglomerate and precipitate, cause the minerals to adhere to the oil-covered bubbles, and the same forces that keep the particles of an oil emulsion dispersed, keep the gangue particles repelled from the bubbles.

Expressed briefly, the theory is as follows:

That oil flotation is an electrostatic process. It is a scientific fact that when a solid particle is suspended in water, the water will form around the particle a contact film which generally possesses an electric charge, the amount and polarity depending upon the nature of the surface of the particle and the electrolyte in which it is suspended. The presence of these charges can be demonstrated by the fact that the particles possessing them will migrate when placed in an electric field. It has been demonstrated that flotable particles have charges of one polarity (positive), and that non-flotable particles have charges of the opposite polarity (negative); that the froth is charged negatively and so attracts the positively charged or flotable minerals, and repels the negatively charged or non-flotable ones. It is this, it is believed, that causes the flotable minerals, galena, sphalerite, etc., to adhere to the froth, and the gangue minerals, silica, etc., to remain in the liquid where they can be discharged as tailing.

In the discussion which followed Mr. Callow's paper Mr. JAMES A. BLOCK said that during the last few months he has tested about fifty samples of ores, etc., the flotation characteristics of which he knew. Some of these were samples of froth and tailings from plants in actual operation, but most of them were samples taken from laboratory tests going on at the plant of the General Engineering Company, and at the station of the U. S. Bureau of Mines, Salt Lake City. In all of these experiments, no real exception to the theory was found. All of the tailing samples produced in actual flotation showed a negative polarity; all of the ores with the so-called "bad gangues," except one or two, showed some positive polarity, and these exceptions could be explained by the extreme dryness and scaliness of the gangue minerals, and all of the froth products showed positive polarity when the samples were fresh. It was often true that after standing some time these froth samples would reverse, especially when they contained considerable oil, but this reversal can be explained if we assume that the oil forms a coating around the particles upon standing.

The question might naturally be asked, "Can these electric charges be put to any practical use?" A machine[1] used to purify clays employs very similar charges on the constituent particles of clay. This machine (see Fig. 1) consists of a revolving cylinder, made of conducting material, such as iron, about half immersed in a clay pulp in a trough. At a distance of about ½ in. from the immersed side of the cylinder is a coarse wire screen. A direct current of about 50 volts was applied to the cylinder and the screen, the cylinder being the anode and the screen the cathode. The cylinder attracted the negatively charged kaolin, while the screen attracted the positively charged pyrite, ferric hydroxide, and aluminium hydroxide. It was found that a product running only about 17 to 20 per cent moisture could be scraped off the upper side of the cylinder, which revolved slowly, while the impurities fell through the screen to the bottom of the trough.

The objection to the process was the high cost of the power needed, which averaged over 30 cents per ton of product, but the process is undoubtedly of interest in that it shows that these electrostatic charges are de-

[1]Transactions of the English Ceramic Society, vol. XII, pp. 36 to 64 (1912-13).

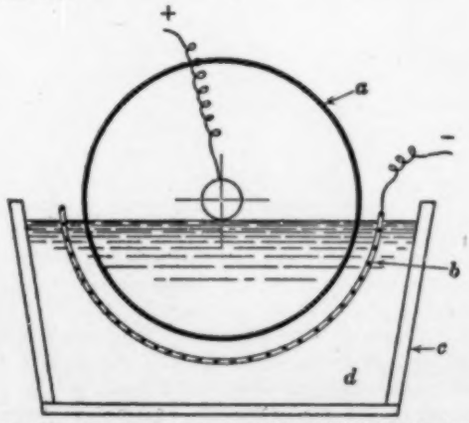


FIG. 1—APPARATUS USED IN PURIFYING CLAYS

a. Revolving cylinder. *b*. Wire screen. *c*. Wood trough. *d*. Clay pulp.

pendable enough to be put to practical uses. It is also of interest in that the polarity of the charges noticed on various minerals checks the results of the experimental work upon which the theory is based.

Mr. OLIVER C. RALSTON of the Bureau of Mines (Salt Lake City Station) said that for some time he had had a growing conviction that possibly the particles dealt with in the flotation of sulphide minerals were acting according to certain laws laid down in colloid chemistry. One of the characteristic things about either suspension or emulsion colloids is that their individual particles are charged with one or the other sign of static electricity. It is known that the amount and signs of these charges can be controlled by the amount and character of electrolytes put into the water in which the colloids are dispersed, and in turn the properties of the colloidal suspension are greatly changed with this change in the electric charges on the particles. The question has been whether the comparatively coarse suspensions of ore can be compared with suspension colloids, or whether they were too coarse to allow the electric charges on the suspended particles of mineral to produce any measurable effect.

"This question has been answered fairly definitely in my laboratory during the past year, for it was found that the electric charges on the particles of what we call 'slimes' and 'fines' are of considerable importance in the control of the rate of settling of such slimes in water. With this much definitely settled, and with a great amount of literature available in the realm of colloid chemistry dealing with the electric charges on particles of such things as quartz, galena and sphalerite, and on oil droplets and air bubbles, it was easy to find a combination of circumstances which might explain flotation from this new viewpoint. At about this time George W. Riter made the prediction that the depths of electrostatics would have to be thoroughly sounded before flotation phenomena could be entirely understood. He seemed to have the idea at that time that the charges on the particles were generated by friction in the flotation machine, in much the same way that a piece of sealing wax becomes charged when rubbed with flannel; but we know from colloid chemistry that these charges of electricity arise whenever two substances of unlike dielectric constants come into contact.

"Further work along the line of controlling the charges of the ore particles, thereby controlling the rate of settling of the particles in water, has convinced me that the charges are of considerable importance, and the results of this work will be reported soon. Mr. Callow told me last spring that he had much the same conception of the matter as I, and had the Mellon Institute in Pittsburgh working on it for some time. Mr. Callow sent a man to the Salt Lake laboratory of the U. S. Bureau of Mines to try out some methods of measurement which I had proposed and which had not been used in Pittsburgh; some of the results are presented in Mr. Callow's paper.

"The knowledge that when flotation conditions are good the froth has a positive charge and the rejected gangue a negative charge, that the oil droplets and the air bubbles are negatively charged, and the sulphide particles positively charged, is highly interesting. We have here electrical effects which are parallel to flotation phenomena, and the conclusion that they are connected with each other seems justifiable. Just exactly how they are connected is not yet clear and may prove difficult of solution.

"It is to be hoped that further work will be done along this line in order that a theory of flotation may be found, which will afford a more rational control of flotation testing, and an extension of the process to the treatment of any kind of minerals."

Radium from Carnotite

On the evening of Friday, Dec. 17, Dr. **Charles L. Parsons,** chief of the Division of Mineral Technology of the Bureau of Mines, lectured at a meeting of the New York Section of the Society of Chemical Industry on "Extraction and Recovery of Radium, Uranium and Vanadium from Carnotite." Dr. Wm. M. Grosvenor, chairman of the Section, presided.

Dr. Parsons' lecture was particularly interesting on account of the exhibit and demonstration of various samples of radium, as well as lantern slides and moving pictures. After the lecture there was some discussion in which representatives of the Standard Chemical Co. participated. The discussion was good natured and dealt especially with the applicability of the process to ores of less than 2 per cent U_3O_8 and the cost and price of radium and the policy of the Bureau of Mines in dispensing with the radium produced in connection with the National Radium Institute (see account below). A leading official of the Memorial Hospital of New York paid a very high tribute to Dr. Parsons' eminent business ability in running the radium factory and gave some information on cancer treatment by radium.

A very full account of the work of the Bureau, done in co-operation with the National Radium Institute, has since appeared in Bulletin 104 of the Bureau of Mines, entitled Extraction and Recovery of Radium, Uranium and Vanadium from Carnotite, by Charles L. Parsons, R. B. Moore, S. C. Lind and O. C. Schaefer. (124 pages, with many illustrations.)

As the general method of treatment, used by the bureau, has already been described in our issue of Dec. 15, 1915 (Vol. XIII, p. 965), we give here only some supplementary information on interesting details.

The capacity of the plant is 3½ tons of ore per day, and up to Oct. 10, 1915, nearly 5 grams of radium element had been extracted and approximately 2½ grams of element delivered in the form of radium bromide of such a degree of purity as was desired by the hospitals concerned (the Howard A. Kelly Hospital

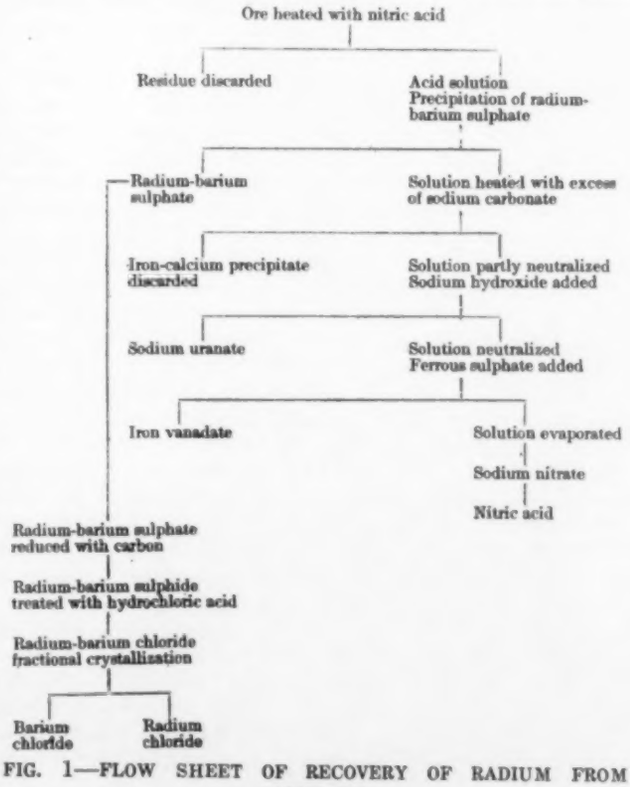


FIG. 1—FLOW SHEET OF RECOVERY OF RADIUM FROM CARNOTITE

in Baltimore and the General Memorial Hospital in New York).

"With ore obtained from Government land or produced at a cost as low as that maintained in the operations of the bureau, it has been shown that the cost of producing radium need not exceed $40,000 per gram and that the extraction of at least 90 per cent of the radium present may be obtained from good-quality ore, such as the bureau has been able to procure.

"The market price of radium for some two or three years has been $120,000 and up per gram of element according to purity. At the time of the beginning of the European war there were 16 to 20 grams of radium contracted for abroad, chiefly in Germany, at prices above $120,000 per gram. Seemingly the war has not affected the market price, but simply the amount produced. As to the selling price of radium in the future, the bureau makes no prediction, but it does not follow that the selling price will decline because of the development of cheaper methods of production. Rather is the case somewhat analogous to the production of gold, diamonds, or any other material that is in steady demand but occurs only in very small quantities. Beyond doubt, the amount of radium in nature is exceedingly small. Other deposits of radium ores may be found, but it is highly improbable that the rarity of uranium ores will ever be greatly modified, so that the price of the finished material will largely depend on the ability to procure the raw material.

"According to the best evidence that the bureau can obtain, and it has investigated the matter carefully, there is not sufficient ore available to maintain for many years even the rate of production of 1914. In this connection it should be remembered that the radium produced by the National Radium Institute is not for sale nor for distribution. The Radium Institute was organized for the purpose of studying the curative properties of radium and not for private gain. The radium produced is being used in two hospitals, the Howard A. Kelly Hospital in Baltimore and the General Memorial Hospital in New York."

According to the agreement between the Bureau of Mines and the National Radium Institute any radium produced in excess of 7 grams of anhydrous radium bromide ($RaBr_2$) from any 1000 tons of carnotite ore may be donated to the bureau for further experiment and study. This excess output will be supplied to the hospitals of the Army, Navy and Public Health Service.

The ore treated by the Bureau contains 2 per cent or more of U_3O_8. In considering the work it must be taken into consideration that the problem is to get one part, by weight of radium, out of 200,000,000 parts of ore. This concentration process becomes possibly only by making use of electrical measuring methods by which radioactive elements can be detected and quantitatively measured in quantities far below the limits of any methods based on other properties.

The method of treatment of the bureau (nitric acid leach) is new. Older methods (leaching with sulphuric acid or hydrochloric acid and fusion with sodium sulphate or sodium carbonate) are discussed in the bulletin. If radium is the principal product desired, the nitric acid leaching method is the most direct and efficient. The cost of the nitric acid process is considerably reduced by the fact that sodium nitrate is recovered with small loss and nitric acid once more made from the sodium nitrate, so that the process is cylic with respect to nitric acid.

In the nitric acid process the radium is at once recovered as a high-grade radium barium sulphate, practically free from silica, and easily treated by improved methods. The process is adapted to recovering either the radium by itself, or the radium, uranium and vanadium, as the radium is obtained first, and from that point all the other products may be discarded without further treatment, if so desired.

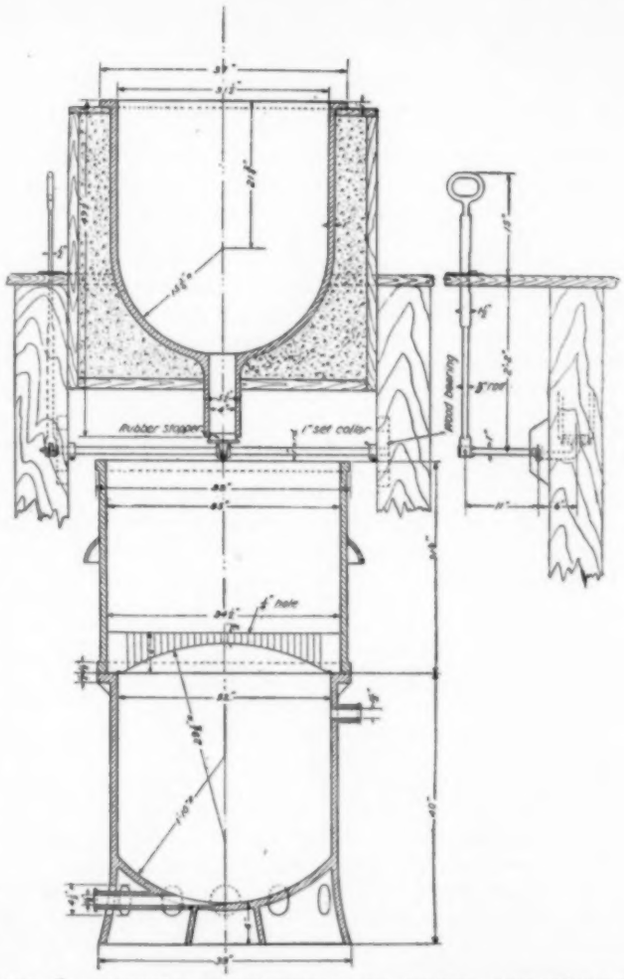


FIG. 2—SECTIONS OF LEACHING POT, FILTER, AND DETAILS OF LEACHING-POT STOPPER

The different steps are each completed in one day, the equipment is not expensive, the extraction and recovery are high, and it is believed that the costs are lower than those with any other process for treating carnotite.

Fig. 1 gives the flow-sheet of the whole process, the different steps of which were already described on page 965 of our issue of Dec. 15, 1915.

Fig. 2 shows sections of the leaching pot, filter, and, at the right, details of the leaching pot stopper. The amount of acid used for leaching is 121 lb. of 100 per cent nitric acid to 500 lb. of ore, the acid being diluted to 38 per cent strength. However, the strength may be varied somewhat, ores high in sulphates requiring stronger acid. The acid is brought near the boiling point by live steam, which has been passed through a baffle in order to eliminate any impurities from the steam. The ore is then run in, being stirred with a wooden paddle during the process. The acid is heated for 15 min. longer, with occasional stirring, and the acid is then run into an earthenware vacuum filter, asbestos filter cloths being used. As much as possible of the sand is held back in the pot by means of a long wooden plug manipulated by hand, and this sand is given an acid wash with acid about one-third of the strength of that used for the first leaching. This sand is then dumped on the filter, and receives two washings with hot distilled water.

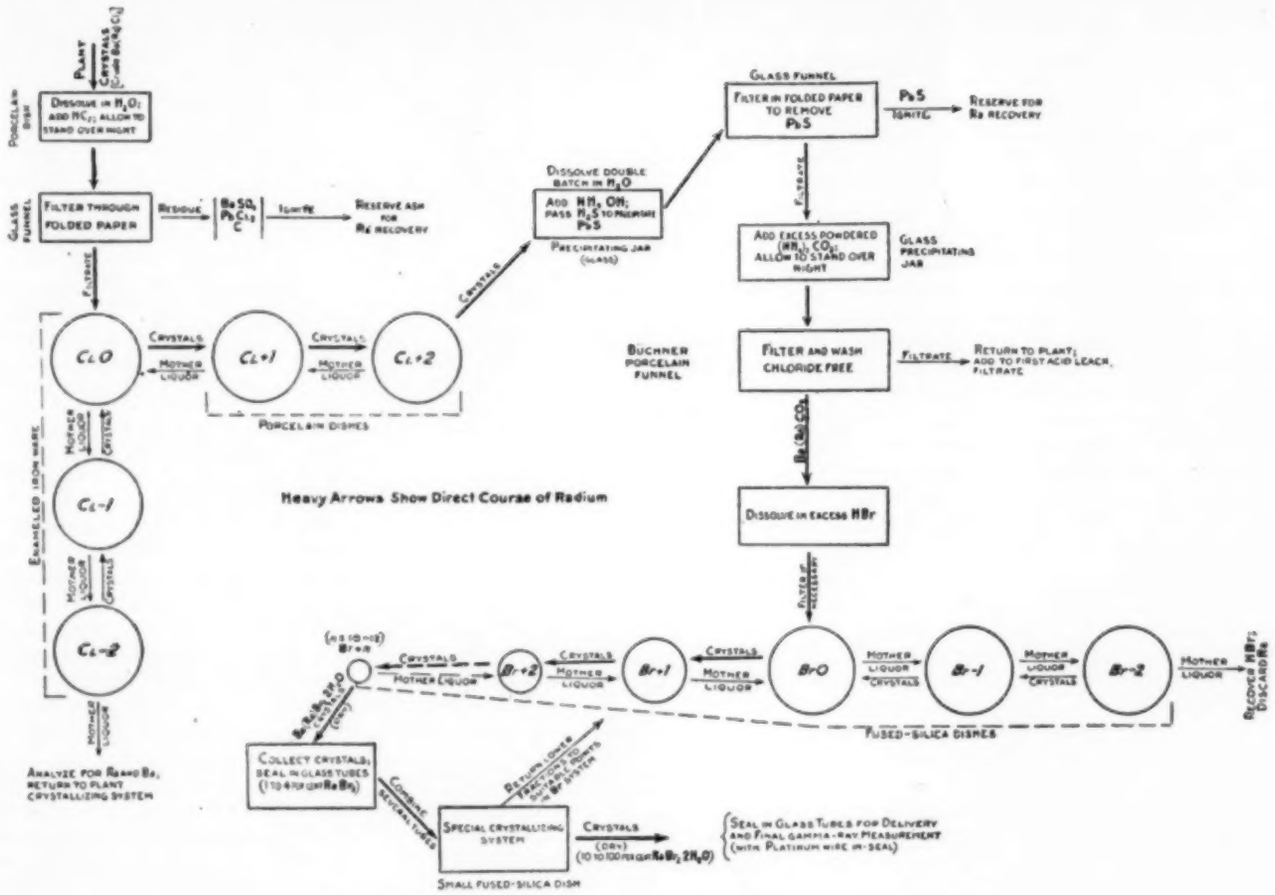

Heavy Arrows Show Direct Course of Radium
Porcelain Dishes
Buchner Porcelain Funnel
Glass Precipitating Jar
Glass Funnel
Enameled Iron Ware
Fused-Silica Dishes
Small Fused-Silica Dish
Filter through folded paper
Filter and wash chloride free
Dissolve in excess HBr
Special crystallizing system
Collect crystals; seal in glass tubes
Analyze for Ra and Ba; return to plant crystallizing system
Seal in Glass Tubes for Delivery and Final Gamma-Ray Measurement (with Platinum wire in seal)
Crystals
Mother Liquor
Filtrate
Residue
Reserve for Ra recovery
Return to plant; add to first acid leach filtrate

FIG. 3—FLOW SHEET OF LABORATORY CRYSTALLIZING SYSTEM

The addition of a small amount of hydrochloric acid to the nitric acid increases the solvent action on the radium. Therefore, if the nitric acid does not contain hydrochloric acid, enough hydrochloric acid is added to make the proportion of concentrated hydrochloric acid to 100 per cent nitric acid, about 2.5 per cent. In other words to each pot, 9 lb. of 28 per cent hydrochloric acid, the strength of the acid available in Denver, is added.

RADIUM REFINING

The first treatment of the precipitated radium-barium sulphate is of great importance, as the whole capacity of the plant, as far as the radium goes, depends upon this first treatment. If the sulphates cannot be handled as rapidly as produced, the actual amount of radium refined each year must necessarily be much below the capacity of the plant itself.

The Bureau of Mines first reduces the radium-barium sulphate to sulphide with charcoal. This is done by mixing the sulphate with about one-fifth its weight of powdered charcoal and heating for 7 or 8 hr. in a graphite crucible in an oil furnace, the temperature being about 800 deg. C. Reduction takes place readily, according to the equation $BaSO_4 + 4C = BaS + 4CO$, with carbon monoxide burning at the spout of the crucible.

After reduction, the sulphide is ground and leached with moderately dilute chemically pure hydrochloric acid in an earthenware pot placed outside of the building. After filtering, the filtrate, which consists of an almost saturated solution of radium-barium chloride, is ready for fractionation.

Radium of any desired degree of purity may be obtained by fractional crystallization of the barium solution, first as chloride and later as bromide. The radium continues to be enriched in the crystal fractions and impoverished in the successive mother liquors.

The salts of most metals, such as iron, aluminum, and vanadium, that may occur with the radium-barium salt as impurities, pass into the mother liquors, and give no difficulty in the enriching radium fractions. Lead is an exception, however, and requires special treatment.

Fig. 3 shows the steps of the concentration system employed in the laboratory. The diagram is self-explanatory. The heavy arrows show the direct course of the radium. Concerning details of the system the reader may be referred to Bulletin 104, which also contains an interesting chapter on radium measurements.

Accidents at Metallurgical Works—In Technical Paper 124 issued by the Bureau of Mines, a compilation of accidents during 1913 and 1914 at metallurgical works in the United States is given. The publication is the work of Mr. Albert H. Fay.

Magnesite.—The annual statement of the U. S. Geological Survey on the production of magnesite in 1914 now available for distribution, shows an increase over 1913 of 17 per cent, 11,293 short tons of crude magnesite valued at $124,223, having been produced or sold and treated in the United States during the year.

Multiple-Unit Furnaces.—Messrs. Eimer and Amend report increasing application of their multiple replaceable unit electric furnaces in the industries. A change in design from the original has made it possible to replace the units very easily in a few minutes by simply unloosening four screws. The furnaces were originally designed for laboratory work, but the replaceable unit idea has since been applied to industrial-size furnaces.

Synopsis of Recent Chemical and Metallurgical Literature

Iron and Steel

Electric Furnace Steel in Canada.—In the discussion of a paper by Dr. A. Stansfield, read before the Montreal Metallurgical Association, Mr. J. E. DAVEY, superintendent of the Canadian Brakespur Co., of Sherbrooke, gave the following description of his company's installation.

"We have in operation at Sherbrooke four 5000-lb. basic-lined, three-phase furnaces, and one 2000-lb. furnace. The dimensions of the 5000-lb. furnaces are 70 in. by 108 in. outside lined down to 34 in. x 72 in. wide, while the smaller furnace has a circular hearth of 36 in. in diameter. All the furnaces are mounted on rockers and tilt to pour their charge. We found it very difficult to construct a roof that would withstand the heat of the electric furnace, but finally laid some ¾-in. water-cooled pipes across the furnace and built a roof of silica bricks resting on these pipes. The electrodes are also water jacketed at the point where they enter the furnace, by four coils of ½-in. pipe. The electrodes are made of 6-in. graphite rods, in 4-ft. lengths, which are threaded and joined together to make one continuous electrode. These are raised and lowered by means of steel wire wound on a drum, passing over a sheave and insulated from the electrode holder, but the electrodes are somewhat unsteady, which causes short-circuiting in the furnace when melting down the scrap. It is intended to install rigid holders so that the electrodes will be raised and lowered in a vertical line. The electrodes for the 1-ton furnace are 4 in. in diameter.

"Each of the larger furnaces is provided with a 500-kw. three-phase transformer which receives the electric current at 6600 volts and delivers it to the furnace either at 40, 70 or 100 volts. The 70-volt connection is usually found the best, as 40 volts would be too low, and with 110 volts the furnace would be difficult to regulate. An ideal arrangement would be to have a transformer to give about 60 volts during the breaking down period, with means for raising this to about 90 volts after the charge has been melted. The transformer is placed as close as possible to the furnace. The cables from the transformer to the top of the furnace are 32 ft. in length; part being flexible to allow of the movement of the electrodes. The cables are composed of fifty No. 6 wires, and the three cables for each furnace are interlaced to reduce inductance losses. The cables are carried overhead, instead of in a conduit, as they are more accessible in case of an accident to the insulation.

"In making a 5000-lb. heat, 2000 to 3000 lb. of scrap, etc., are charged through the doors, which are in the roof, and when this material is melted the remainder of the charge is added to make a total of 5250 lb., thus allowing for the loss of about 4 per cent of the metal. The melting occupies from 4½ to 5 hr., and is followed by refining. A considerable reduction of both carbon and sulphur is effected during the melting, the sulphur being easily reduced to 0.04 per cent or 0.03 per cent, and by means of an oxidizing slag the phosphorus is reduced to 0.02 per cent. The specification limits the carbon to 0.55 per cent, the silicon to 0.30 per cent and the sulphur and phosphorus to 0.05 per cent each, while the manganese must be between 0.40 per cent and 1.0 per cent. The power consumption per ton of molten steel in the ladle is about 600 kw.-hr.; the cost of refractories is about $2.60, and of electrodes $2.70 per ton of steel. The molten steel from the furnace is poured into a ladle [illegible] the ladle is carried by a 4-ton trolley crane to a turntable, on which are placed enough ingot molds to take the contents of one furnace.

"Although these furnaces, which have been designed and constructed by the Brakeshoe Company, do very satisfactory work, there will be installed by the end of October a 2-ton Snyder single-phase furnace. This is guaranteed to give ten heats of two tons each in 24 hr. and is expected to give twelve heats of 5000 lb. in 24 hr. This furnace has one 5-in. graphite electrode entering the roof, and a steel column passing through the basic lining and the furnace bottom. The electric current is supplied at 220 volts from a substation placed on the other side of a wall. The cables pass through a conduit to the electrode holders. The electric arc varies from 14 to 18 in. in length."

Gold and Silver

Treatment of Antimonial Gold Ore.—According to the *South African Mining Journal* of Oct. 9, 1915, the problem of successfully treating antimonial gold ore on the Murchison Range has been solved by a process introduced by McArthur Forrest & Company. In brief the process consists in dissolving the antimony sulphide from the crushed ore with a hot solution of caustic soda, recovering this constituent as a saleable product and cyaniding the residue for gold. At the United Jack property the process is as follows: The ore is crushed in ball-mills to 30-mesh and delivered to vats 23 ft. in diameter and 8 ft. deep. Caustic soda solution is then pumped on the pulp and leaching is conducted as in cyanidation. The resulting solution is treated with carbonic acid gas from a lime kiln, which precipitates the antimony sulphide in an amorphous condition. The mixture is filter-pressed and the resulting cake marketed. The solution is returned for further use. The sand residue, containing gold is then cyanided in the usual manner. The original ore is reported to contain gold to the value of $7 to $8 per ton and about 12½ per cent antimony sulphide.

Copper

Pyrite Smelting at Mount Lyell.—In his address as president of the Australasian Institute of Mining Engineers, Mr. ROBERT STICHT has given the most interesting and instructive account yet published of the smelting practice at Mount Lyell. The address appears in full in the Society's *Proceedings*, No. 19. Introducing his remarks by saying that pyritic smelting is something "in which doing is decidedly more difficult than speculating on the 'underlying principles,'" the author proceeds to outline "how it is done" rather than "why it works." We give herewith his "mental picture" of the process, leaving some other details for further presentation.

"In order to understand our method, it is necessary to form a correct mental picture of the inner appearance and working condition of the pyrite furnace. Its most salient feature is a pronounced openness or porosity of the masses in action, together with a total lack of stratification or bedding, yet with an even distribution of the various components of the charge throughout each horizon. There is no layer-like arrangement of the different materials, and no alternation of charge and fuel. There is no reducing zone, but an oxidizing one, restricted, like the first-mentioned, to the lower regions of the column. In distinction from the ordinary furnace, in which a distinct reducing zone precedes the slag-forming zone, the oxidizing and the slag-forming zone are identical. From top to bottom, however, there is one element of the charge present everywhere, which forms the skeleton, so to say, of the

furnace-body in action. This element is the *free silica* in charge. The lumps of quartzite, etc., are present in excess of the quantity required for the momentary furnace action, and they build up a honeycombed or vesicular column, reaching from the crucible of the furnace upward to the stock-line, or top of column. This silica skeleton is a necessity. Without it pyrite smelting is unthinkable. It forms a physical structure like an aggregate of cells, each cell in the heart of the furnace forming a little Bessemer vessel with silicous walls. The wasting away of the material of the walls with scorification does not unsettle the fixity of the column, for the lumps of silica are constantly being replenished by new ones settling down from above.

"More particularly, the furnace column may be divided into three regions—an upper preparatory one; a central one, where the entire chemical activity takes place; and a lower one, extending from above the tuyeres to the bottom of crucible, in which no action takes place. The first contains the charge undergoing only primary chemical and physical changes, the second is the seat of oxidation and scorification, and the last only forms a passageway and receptacle for the molten products. The most important region is the central one. It is situated some distance above the tuyeres, much higher up in the shaft than the slag-forming zone of the ordinary furnace, and fluctuates up and down in position in accordance with the variations in the pneumatic conditions within the shaft, its location being principally decided by the volume of blast blown in. To characterize it as that part of the shaft or column in which the vital functions of the pyrite furnace are in play, I have designated it the 'focus.' The three zones doubtless merge into each other, the lower preparatory zone and the upper reaches of the focus probably not being clearly defined. The lower margin of the focus, however, is probably sharply divided from the basal region. The furnace activity is all upwards, not downwards, the sinking of the fused products and the silica excepted.

"The preparatory zone is probably not so high as in the ordinary furnace, and has the usual functions of driving out moisture and preheating the masses by means of the escaping gases. It has also a very special one, in the case of the pyrites, without which the further furnace action would not take place as it does. This consists in the sublimation of the volatile sulphur atom out of the pyrites, at the top of the zone, and the fusion and superheating, lower down, of the residuum, the composition of which, through the loss of about three-sevenths of the total sulphur, is practicaly that of pyrrhotite, Fe_7S_8.

"In this uppermost zone no *chemical* action affecting the ores on charge takes place, and heat is only being absorbed, but it appears that within this zone a chemical interchange is enacted, accompanied by the evolution of a small amount of heat, between the heated sulphur dioxide, coming up from below, and the coke fed in, as soon as the latter sinks to a horizon where it is hot enough. If the amount of coke is small, it probably does not survive this reaction, and no coke gets down to the focus. Should the amount of coke be large, then some may reach the focus, but it will necessarily interfere with the reactions there, and spoil the pyritic work. A small amount of coke on charge is, however, still a *sine qua non*, and seems required to support the work going on in the preparatory zone, even though its thermal effect is low, under the circumstances. As soon as the melting point of the pyrrhotite residuum is reached in this zone this substance melts, and we now have the phenomenon of an incandescent, honeycombed mass of silicious material in lumps, in the interstices of which aggregate the molten sulphide courses downward, towards the focus, in a heavy shower.

"The atmosphere in the preparatory region consists only of the neutral gases forming the products of combustion from the focus, and the CO_2 out of the limestone and coke, and is heavily charged with sulphur vapor, derived from the disintegrating sulphide of iron and the reaction between coke and sulphur dioxide. By far the chief constituent is nitrogen; then follow sulphur dioxide and carbon dioxide. There is, or should be, an absence of oxygen, and consequently no roasting or other oxidizing action can take place in this zone. In the lower portion of the preparatory zone the limestone may be calcined, but this is more likely to be completed in the upper regions of the focus.

"Physically, the preparatory zone represents a mere jumble of charge constituents undergoing certain changes. Yet the distribution of the constituents should be homogeneous and properly balanced, otherwise the furnace will not work well. Occasionally it may be necessary temporarily to disturb this regularity of mixture by placing more pyrites, or more silica, or more slag, here or there, for the purpose of curing irregularities in the descent of the masses, to relieve the beginning formation of accretions, or to close blowholes, etc., but even this treatment, though of a curative nature, cannot be persisted in too long."

Recent Chemical and Metallurgical Patents

Detinning

Electrolytic Detinning.—In the alkaline electrolytic detinning process it has heretofore not been possible to construct an apparatus made up of a plurality of cells connected together owing to the difficulty of circulating the electrolyte and of preventing the supply pipes from becoming plated with tin, thereby reducing their inside diameter. The process has, therefore, been carried out in single cells. According to a patent of Dr. HANS GOLDSCHMIDT of Essen, Germany (assigned to Goldschmidt Detinning Company of New York), it is now possible to carry out this process in a large number of cells, connected in series with each other, to maintain a good circulation of electrolyte, and also to prevent the circulating pipes from becoming plated with tin. This is accomplished, as shown in Figs. 1 and 2, by providing a reservoir tank 1 above the electrolytic vats and connecting this tank with the vats by a series of pipes as shown. The pipes are made of decreasing diameter as they approach the vats, as this has been found to be very important in providing a good flow. The ends 27 of the pipes, which supply the fresh solution from the high tank, do not touch the liquid and are made of rubber, thus preventing any possibility of becoming obstructed due to tin coating. Each cell has its own

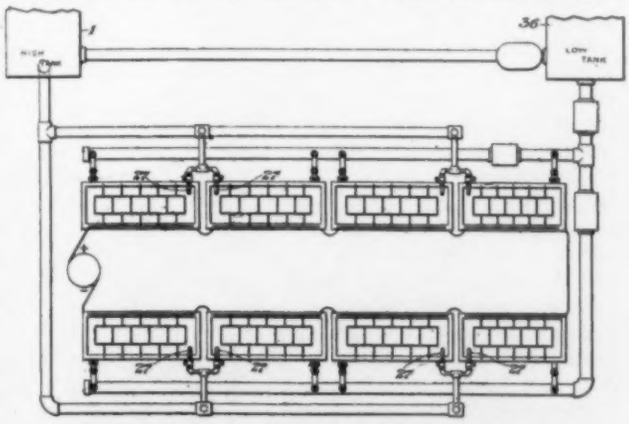


FIG. 1—ELECTROLYTIC DETINNING SYSTEM

inlet and outlet pipe directly connected to the large tanks. The lower tank 36 receives the outlet, and it is pumped from here to the upper tank.

A temperature of 70 to 80 deg. C. is maintained, and a high current density, 800 to 1000 amp. per square foot,

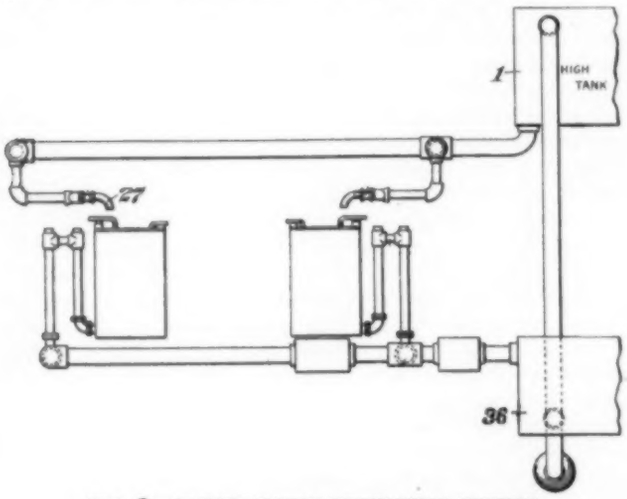


FIG. 2—ELECTROLYTIC DETINNING SYSTEM

is used. The electrolytic vats are made of cast iron encased in wood and form the cathodes. The anodes are usually in the form of a basket containing the material to be detinned. The solution used for the electrolyte is a mixture of stannate of soda and caustic soda with an excess of caustic soda. Carbonic acid from the air is taken up, forming sodium carbonate. The process is continuous as the electrolyte is regenerated, as it is circulated through the system (1,160,400 and 1,160,401, Nov. 16, 1915).

Detinning Process.—A rather novel process for removing metallic coverings from other metals, such as tin from tinned steel scrap, is patented by Dr. HANS FOERSTERLING and HERBEPT PHILIPP of Perth Amboy, N. J. (assigned to Roessler & Hasslacher Chemical Company). This process depends on the alloying properties of alkali metals with other metals, such as tin, and as

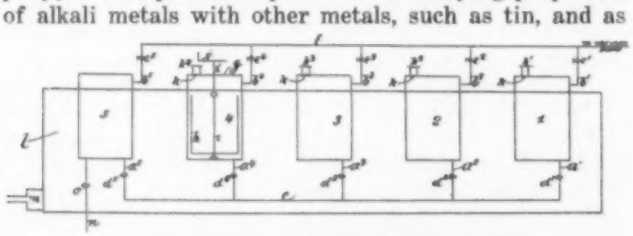


FIG. 3—SODIUM PROCESS OF DETINNING

applied to detinning it consists in treating the tin scrap in a heated container with molten sodium. In Fig. 3, numbers 1, 2, 3, 4 and 5 represent iron vessels provided with drain pipes a_1, a_2, a_3, a_4, a_5 and vacuum pipes b_1, b_2, b_3, b_4, b_5. A common pipe C connects all the drain pipes and a common vacuum pipe f leading to a vacuum pump connects the separate vacuum pipes. Stop cocks are provided on all pipes. In conducting the process all the vessels are placed in one furnace as a unit and heated by the gas burner m. Vessels 1, 2, 3 and 4 have openings k in the top, and vessel 4 has a removable cover and a revolving basket h inside, rotated by shaft i and wheel j. When the process is running, vessel 3 will contain a 25 per cent tin-sodium alloy, vessel 2 a 5 per cent tin-sodium alloy, and vessel 1 pure sodium. The temperature is maintained at about 400 deg. C. The basket h is filled with tin scrap and the cover put on. The vessel is then evacuated, the vacuum shut off and the alloy from vessel 3 allowed to run in. The basket is then rotated, when the tin from the scrap alloys with the sodium very quickly. A vacuum is then created in 3 and the alloy run back again. In like manner the tin scrap is treated with the weaker tin-sodium alloy from 2, and then with the pure sodium from 1, after which the scrap is removed and fresh scrap put in. After the alloy in 3 has reached 50 per cent it is conveyed into vessel 5, when it may be led away through pipe n for distilling off the sodium and recovering the tin. The alloy from 2 is run into 3, the alloy from 1 is run into 2, and fresh sodium placed in 1. Thus a continuous process is formed. The rapidity of action of the sodium is noteworthy. The following table shows the time required:

Alloy	Per Cent Tin in Scrap Start	Per Cent Tin in Scrap Finish	Time, Min.
60 tin 40 sodium	1.8	0.44	10
50 tin 50 sodium	1.8	0.22	10
40 tin 60 sodium	1.8	0.13	10
30 tin 70 sodium	1.8	0.018	10

"The residual iron when removed from vessel 4 still contains traces of sodium, and when molten shows a superior quality over ordinary iron" (1,160,590, Nov. 16, 1915).

Electrolytic Furnaces

Manufacture of Amides, Cyanamides and Cyanides.—A continuous process for the electrolytic production of amides, cyanamides and cyanides is patented by Edgar A. Ashcroft of London, England. This process is based on his previous patent (801,199, Oct. 10, 1905). The reactions taken advantage of are those of ammonia and metallic sodium forming sodium amide between 120 and 440 deg. C. ($NH_3 + Na = NaNH_2 + H$), sodium amide and carbon forming sodium cyanide between 350 and 600 deg. C. ($2NaNH_2 + C = Na_2N_2C + H_4$), and sodium cyanamide and carbon forming sodium cyanide at about 800 deg. C. ($Na_2N_2C + C = 2NaCN$). The apparatus consists of two cells, as shown in Fig. 4. In the first cell a sodium chloride or hydroxide electrolyte is used and a sodium lead alloy is produced. This sodium lead alloy is then transferred to the second cell, where it is used as the anode for the production of amide, cyanamide or cyanide. The electrolyte used in the second cell is made up of the product which it is desired to obtain. For instance, for producing sodium amide, an electrolyte of sodium amide is used, and as ammonia gas is conducted in more sodium amide is formed, which continuously augments the electrolyte. The sodium amide is tapped from time to time as produced. It is possible in making sodium amide to use a solution of the raw materials in the first cell and a mercury cathode, but in making sodium cyanamide a mercury cathode is not suitable. In making cyanamide the procedure is the same as in making amides except that carbon is added to the electrolyte in the second cell to react with the amides as

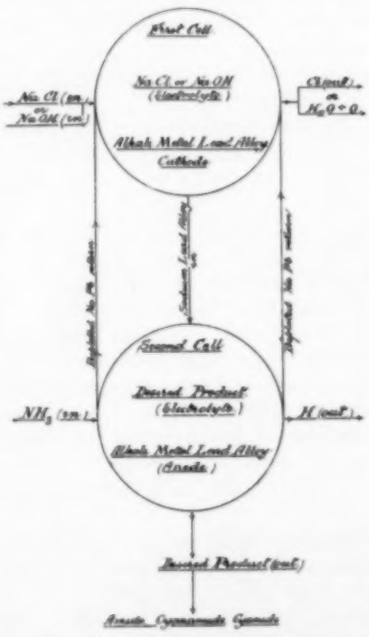


FIG. 4—DIAGRAM OF PROCESS

fast as formed and the higher temperature is needed. In making cyanide the temperature is raised further to about 800 deg. and an electrolyte of cyanide is used. The depleted alloy is returned to the first cell for reuse. It is claimed that the process is more economical and efficient than previous processes (1,163,498, Dec. 7, 1915).

Manufacture of Sodium Alloys.—In electrolyzing fused caustic alkalies for producing alkali metals using insoluble anodes, it has generally only been possible to obtain half the current efficiency owing to one molecule of the alkali out of every two formed being required to decompose the water formed by the reaction, $2NaOH = 2Na + H_2O + O$. In order to increase this efficiency, EDGAR A. ASHCROFT of London, England, proposes to use a higher temperature and provide for the escape of oxygen and steam, instead of oxygen and hydrogen. Further features of the process are the maintaining of only a thin layer of the alkali hydroxide above the heavy metal alloy cathode, and the agitation of the electrolyte and alloy cathode by a pump and magnetic stirring device and the use of a higher current density at the anode. The process is carried out in a copper-lined or nickel-lined cell, either singly or in conjunction with a secondary cell, for the production of amides, cyanamides, or cyanides. An annular plate of nickel is used as anode. The temperature used is between 400 and 500 deg. C., as compared with 330 deg. in previous processes, and efficiencies of from 60 to 80 per cent are claimed. The process is suitable for employment in large units and currents up to 10,000 amp. may be used in a double cell (1,161,585, Nov. 23, 1915).

Apparatus for the Manufacture of Sodium and Sodium Alloys.—In processes for making sodium by electrolysis of a fused mixture of sodium and potassium chlorides with a molten lead cathode, it has heretofore been difficult to produce sodium free from potassium, due to potassium alloying with the sodium and lead in the primary cell and passing into the final product. In an apparatus patented by EDGAR A. ASHCROFT of London, England, these difficulties are claimed to have been overcome. Reversible reactions, such as the following, have been found to occur between the electrolyte and alloy in the primary cell and between produced alkali metal and electrolyte in the second cell:

$$NaNH_2 + K \leftrightarrows KNH_2 + Na$$
$$NaCl + K \rightleftarrows KCl + Na$$

It has been found that these reactions may be controlled in such a way as to produce sodium which is practically free from potassium. The second reaction takes place more readily from left to right, and this fact may be made use of by electrolyzing a mixture of 40 per cent sodium chloride and 60 per cent potassium chloride and forming the lead-sodium-potassium alloy as usual. This alloy is then separately heated with a fresh electrolyte of NaCl, which removes the K from the alloy. The chlorides are then conducted to the first cell and the alloy free from K to the second cell for further use. This "triple" procedure has been found unnecessary, however, in most practical cases, and the following procedure has given a good product: The electrolyte in the first cell, consisting of sodium chloride with 2½ per cent or less of potassium chloride, is kept in whirling motion by creating a magnetic field around the cell by a metallic conductor. The anode is a hollow graphite cylinder placed at the center so made as to carry off the chlorine generated which is forced toward the center by the whirling motion. The first or primary cell is lined with a basic insulating lining as a temperature of about 750 deg. C. must be maintained (molecular proportions of sodium and potassium chlorides melt at 670 deg. C.). The whole apparatus is cast in one piece of steel, and another feature is the addition of an intermediate receptacle for the alloy connected by weirs and overflows to both cells so that changes in bulk will not affect the continuous operation of the cells. The alloy flows continuously from the first cell into the intermediate receptacle, and is pumped or flows from there into the second cell as required, a constant level being always maintained. Passages are also provided for conducting the depleted alloy back to the first cell (1,159,154, Nov. 2, 1915).

Manufacture of Cyanides and Cyanamides.—An electrolytic process for manufacturing cyanides and cyanamides is patented by CHARLES E. ACKER of Ossining, N. Y. (assigned to Nitrogen Company of New York). In an electrolytic furnace 1 (Fig. 5) of cast

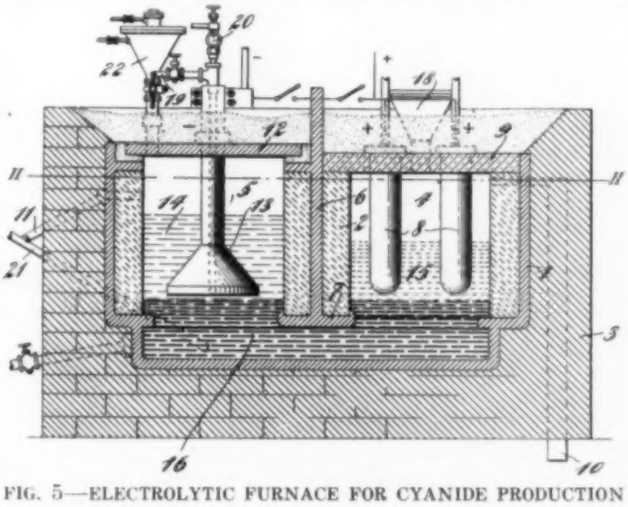


FIG. 5—ELECTROLYTIC FURNACE FOR CYANIDE PRODUCTION

iron or steel there are a primary chamber 4 and a secondary chamber 5, partially separated from each other by the wall 2 of refractory material. The anodes 8 are of carbon and the cathode 13 is of iron, nickel or carbon. The electrodes are adjustable. The chambers 4 and 5 are closed at the top by covers 9 and 12. The primary electrolyte consists of molten barium chloride with a small amount of sodium chloride at the beginning of the operation. The secondary electrolyte consists of molten sodium cyanide. Both electrolytes rest on a bath of molten lead shown at 16. In starting the process the electrodes are raised slightly above the surface of the lead before turning on the current. Barium and sodium are liberated in the primary cell, an alloy of lead, barium and sodium being formed. This alloy is put into circulation by means of a centrifugal pump and soon becomes homogeneous and of uniform temperature. In the secondary chamber, in which the lead-barium sodium alloy is the anode, the barium combines with CN ions set free by electrolysis, forming barium cyanide. Metallic sodium is liberated at the cathode, which reacts with the barium cyanide, forming sodium cyanide and barium. Charcoal is introduced through hopper 22, having first been exhausted of its air by a pump. The barium set free by the metallic sodium combines with the carbon forming barium carbide, which remains suspended in the cyanide. Either ammonia or atmospheric nitrogen is then introduced through pipe 19, passing down through cathode 13, or through pipe 21, directly to the molten metal. The latter pipe is used when it is desired to preheat the nitrogen-containing gas by means of the molten metal, as in the case of air. The nitrogen then combines with the barium carbide forming barium cyanide, and thus a cyclic process is formed. As soon as sufficient barium has been added to the process through the primary cell it is unnecessary to add more, and the primary electro-

lyte may consist only of sodium chloride. When enough cyanide has been formed the charge is clarified by stopping the flow of gas first, then the liberation of sodium, and allowing the mass to settle. Barium carbide with the excess carbon will settle and the cyanide may be tapped off, or the sodium supply may be cut off and the product drawn off as a mixture of barium and sodium cyanides (1,160,811, Nov. 16, 1915).

Blower for Flotation and Smelter Work

A blower which meets successfully the requirements of flotation and smelter service in which the air supply is so essential a consideration, is shown in the accompanying illustrations. This blower is known as the Boston type blower and is made in various sizes, according to capacity and pressure desired, by the Connersville

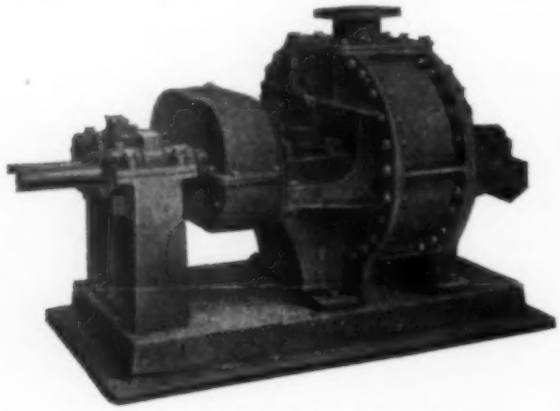


FIG. 1—BOSTON TYPE BLOWER

Blower Company, Connersville, Ind. It is rationally designed for high-pressure work and has found much application in smelter and flotation work. Fig. 1 shows the blower complete and Fig. 2 shows the interior construction and principle of its operation. The rotating parts, called impellers, do not come in contact with each other or with the case at any point in the revolution. They are finished all over with at least two cuts and are separated by accurate clearances so that there is no internal contact or wear and no need for internal lubrication. The impellers are symmetrical, carefully balanced and heavily ribbed internally. They are pressed and keyed onto the shafts and may be operated at high speeds without vibration.

The losses of a positive pressure blower are confined to mechanical friction and slippage. The slippage loss is determined by the pressure, regardless of the speed. Therefore, high speed is desirable because it means high efficiency by reason of the low per cent of slippage loss. Blowers not so well made cannot safely be operated at economical speeds.

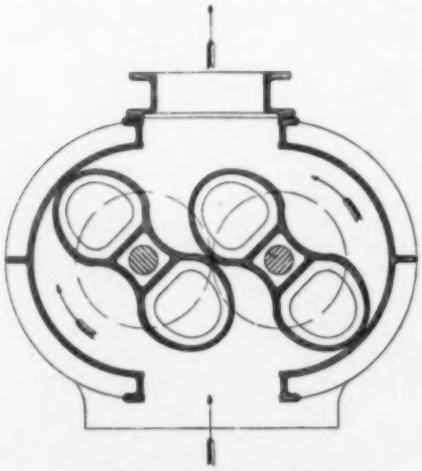


FIG. 2—INTERIOR OF BLOWER

One element contributing to the serviceability and efficiency of this high pressure blower is the type of gear which is used exclusively. This is the epicycloidal radial flank gear whose tooth curve is generated by a rolling circle the diameter of which is one-half the pitch diameter of the gear. The tooth has minimum curvature and maximum working surface. Its action approaches rolling contact.

Special cutters are used, each designed for only one pitch diameter and one number of teeth. This practice and the fact that the tooth is thickest at the pitch line and can be accurately calipered at both ends, gives a two and three-point working contact as against the one-point contact of ordinary gears. The latter necessarily transmit their power in a series of small impulses or blows at once noisy and destructive. The gears operate in a bath of oil fully protected from dust or injury, being contained in a cast iron housing apart from the blower case. They are supported by bearings on both sides. This construction eliminates vibration, insures correct meshing and a contact the full width of the gear face.

The Latest Improvements in Brinell Hardness Testing Machines

The Scientific Materials Company of Pittsburgh has made extensive studies and numerous practical tests with all existing types of Brinell metal hardness testing machines and has succeeded in developing a much improved model.

The most important drawback in the design of old styles of such apparatus (most of which work on the hydraulic principle) is that the transmission and measurement of pressure is not constant, thus introducing serious errors in the determinations, which increase with the age of the machine. Ground cylinders for pressure transmission can never produce that constancy which is essential in a first-class testing machine.

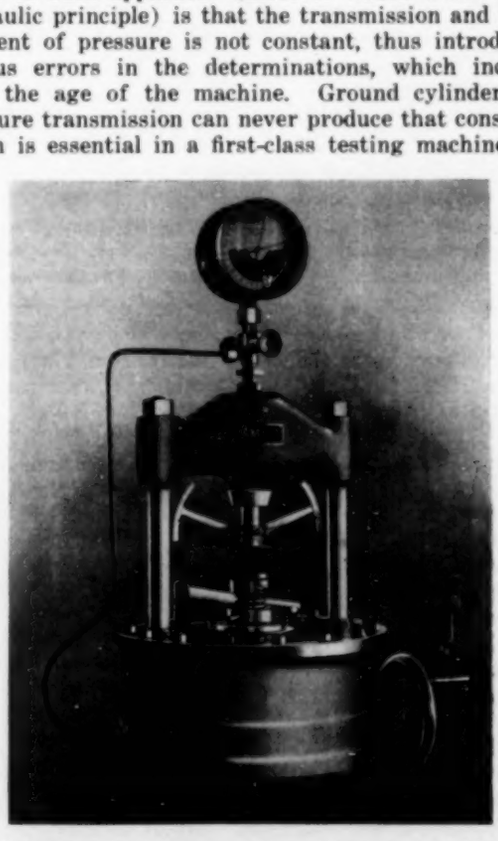


FIG. 1—SCIMATCO-BRINELL HARDNESS TESTING MACHINE, 3000-KG. CAPACITY, DIRECT-READING FOR DEPTH OF BRINELL INDENTATIONS

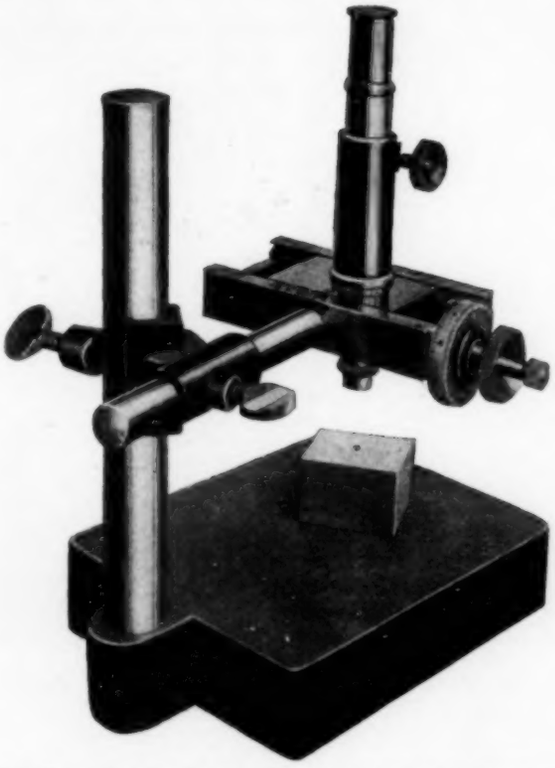


FIG. 2—SCIMATCO-BRINELL MICROSCOPE FOR DIAMETER MEASUREMENTS, DIRECT READING

Another important point established in a long series of tests run in the metallographic laboratory of the Scientific Materials Company and confirmed by separate research work along the same lines at the U. S. Bureau of Standards is that the method of determining the hardness of metals by measuring the depth of indentation of the Brinell ball is far more accurate than the method of estimating the hardness from the diameter of such indentations. Furthermore, the depth measurement permits the detection of small differences in hardness and allows the direct reading of the correct result, eliminating all tedious calculations.

The Scientific Materials Company has succeeded in embodying a direct-reading depth gauge in the design of its new Brinell machine and has eliminated the serious errors of old types, by using the only correct

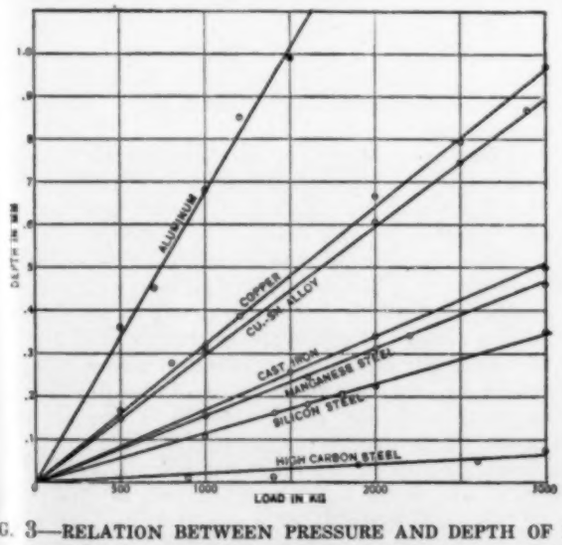


FIG. 3—RELATION BETWEEN PRESSURE AND DEPTH OF INDENTATION ON METALS OF DIFFERENT HARDNESS

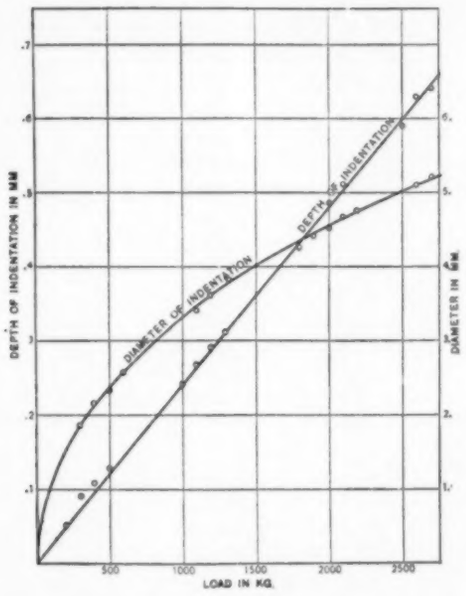


FIG. 4—RELATION BETWEEN DIAMETER AND DEPTH OF INDENTATIONS ON ONE BAR OF BESSEMER STEEL, AT VARIOUS PRESSURES

method of hydraulic pressure transmission. As will be understood from Fig. 1, the hydraulic pressure is produced in the bottom vessel of the machine by the turning of a spindle. This permits a steady and finely regulated increase and decrease of the pressure and represents a considerable improvement over the old pumping devices which naturally raise the pressure in jerks.

The hydraulic pressure is then transmitted by a metallic vessel filled with liquid which acts similarly to the closed diaphragm vessel of an aneroid barometer. This is the most accurate and only constant method of hydraulic pressure transmission known to modern science. By this "hydraulic dynamometer" all variations in friction are automatically eliminated. Since there is no friction in the Scimatco machine, there can be no friction variations, and thus the most serious error of the old types of similar machines can never occur in this improved hydraulic press. A very simple and effective controlling device is provided to check and regulate the volume of liquid in the dynamometer vessel, so that this feature is also at all times under full control.

The direct-reading depth gauge carried by the Scimatco-Brinell machine is designed and constructed in such a manner that it will give correct indications to 1/100 of a millimeter, independent of the shape of the test piece. The stage, on which the specimen rests, is automatically levelling so that pieces of irregular shape can be readily investigated. The pressure gauge embodied in the Scimatco machine consists of two separately acting, but self-contained, pressure-measuring units: one is used as "working gauge" in the routine determinations and the other acts as "standard" to check the working gauge from time to time, say, once every month. The connection between these two gauge units and the pressure vessel can be conveniently made and interrupted by spindle valves.

The dial of the gauge is uniformly graduated in equal units, and not in kilograms. A calibration certificate is furnished with each machine, showing the pressure equivalents of these units. The uniformly and finely divided gauge is much easier to read than ordinary pressure gauges which must naturally be unevenly divided, and, in case the calibration of the "working

gauge" should show deviations after a certain time, it will only be necessary to change the calibration certificate.

Should it be necessary to measure the diameter of the indentations, for the purpose of comparing results with those of other investigators who have no means for ascertaining the depth of Brinell indentations, it can be done after the specimen is removed from the machine. For the purpose of very accurate diameter measurement, the Scientific Materials Company has constructed a special measuring microscope (Fig. 2) mounted by universal clamp on a very substantial support and base, permitting convenient diameter measurements on heavy and irregular shaped metal pieces by direct-reading scale.

The eye-piece of this precision instrument carries cross hairs which are focused against the indentation. Then the joint of the cross hairs is set against one side of the indentation, the position of the scale and graduated micrometer screw noted, then moved to the other side of the indentation and the position again carefully noted. The difference between the two positions gives the diameter of the indentation with an accuracy of 1/1000 millimeter.

Figs. 3 and 4, taken from a recent publication of the U. S. Bureau of Standards, will illustrate the superiority of the method of determining hardness by measuring the depth of indentations instead of diameters. Fig. 3 shows plainly that in testing the hardness of soft, medium, and hard metals the depth of indentation at various pressures forms a straight line, thus proving beyond doubt the consistency and accuracy of results.

Fig. 4 shows that on one and the same piece of metal the diameters at different loads are not in proportion, while the depth measurements show a straight line.

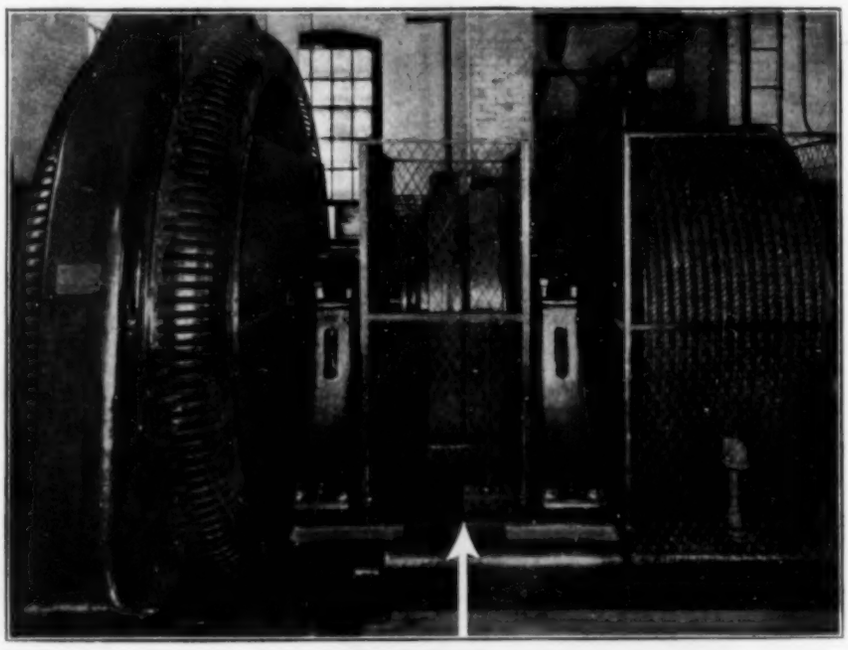


FIG. 1—MAGNETIC CLUTCH-BRAKE USED ON MILL DRIVE IN PLANT OF BOSTON WOVEN HOSE & RUBBER CO.

Motor is of synchronous type—650 hp. Arrow points to clutch-brake

Magnetic Clutch-Brake in Rubber Mill

BY C. B. FERGUSON

In the plant of the Boston Woven Hose & Rubber Company it was planned to drive the rubber mills in groups from a single shaft by means of a synchronous motor of 650 hp. The use of a synchronous type motor was desirable because of its characteristic of raising the power factor of the power system. Two problems were presented—one the safety stop feature of every rubber mill, the other the starting of the synchronous motor.

The safety stop feature is an important one in the rubber mill room of to-day. The easy means of throwing off the driving power and applying a powerful brake has saved many hands, arms, and even lives. This feature is provided by the Cutler-Hammer magnetic clutch-brake installed and shown in the illustration Fig. 1.

The starting of the synchronous motor is another problem. With this type of motor means of easy and gradual acceleration of a considerable load must be provided to prevent the load from being thrown on the motor too suddenly and pulling the motor out of step. Since the station supplying power for this synchronous motor was already loaded up to its capacity it was important to provide means of properly accelerating the motor. This means consisted of the magnetic clutch-brake referred to above.

The equipment consists of a synchronous motor mounted on a common sub-base with a rope sheave, the magnetic clutch-brake furnishing the connection between the motor and the rope sheave. The motor is rated at 650 hp., runs at 240 r.p.m., and under normal conditions carries practically full load when the mill line is operating. Safety switches are installed in the mill room and a slack cable switch in the motor room to stop the motor and line shaft in case of the driving rope breaking.

The chief points of this installation are, therefore, safety in the mill room is provided by safety switches which stop the mills in case of accident; gradual acceleration of the synchronous motor is obtained by means of automatic rheostat and clutch-brake; accidents are prevented because the motor cannot be started by closing one of the safety switches, the station operator must do it by closing the panel switch; the slack cable switch in the motor room automatically disengages the clutch and causes the brake to be applied in case the driving ropes break; a stopping push button is provided on the control panel for ordinary stops when the clutch is simply released but the brakes not applied, thus saving unnecessary wear.

With reference to details of the clutch-brake or clutch-brake accelerator, as sometimes called, it should be noted that the clutch portion consists of a solid circular steel casting, mounted on the driving or motor end of the shaft. This member of the clutch carries embedded in its periphery a single cylindrical magnetizing coil, the terminals of which are brought out to an ordinary pair of slip rings. Mounted concentrically with the coil and the outer periphery, is supplied an adjustable ring of friction so that the metal faces of the driving and driven member do not come directly in contact with each other. This also serves the purpose of providing a permanent air-gap in the magnetic circuit. The other member of the clutch, keyed to the driven or rope pulley side of the shaft consists of a circular steel armature secured to the hub by means of a flat circular spring plate, so that when the magnetizing coil is energized, this circular armature is drawn toward the driving

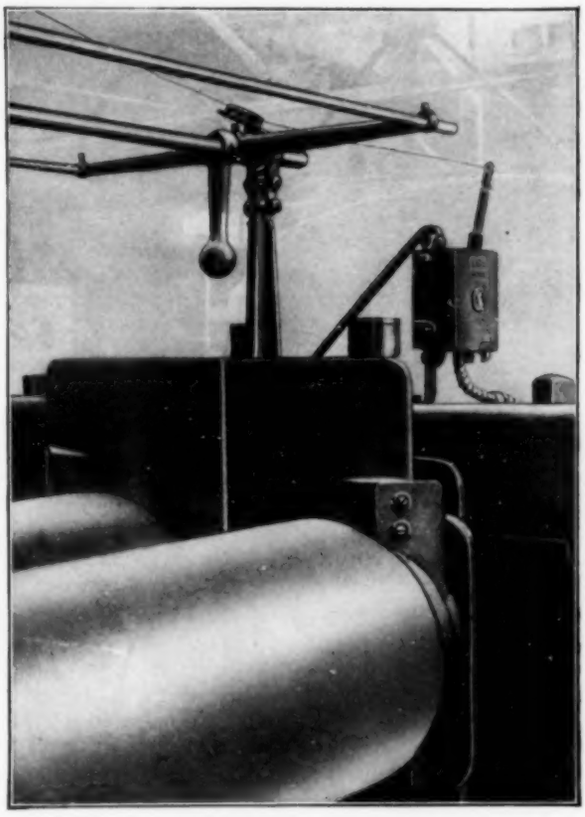


FIG. 2—SAFETY SWITCH

It is tripped when rod on either side of rolls (running parallel with them) is pulled down. This throws out the clutch and causes brake to be applied instantly.

member of the clutch, the motion along the shaft being accomplished by a slight dishing of the spring plate upon which it is mounted.

It will be noted from the description just given of the clutch portion of this equipment, that this particular form of clutch possesses the advantage of instant disconnection in case of interruption of the magnetizing current and does not require a portion of a revolution for its disengagement. Another feature easily understood from the above description is that with this form of clutch, there is no tendency for it to either release itself or to lock itself due to the mechanical forces exerted as it comes up to speed and while it is running at full speed.

Secured to the hub of the driven member is an ordinary brake pulley, over which there is carried a band brake of the locking type, automatically released by a solenoid mounted on a separate stand in proper relation to the brake beam. The clutch itself is 54 in. in diameter.

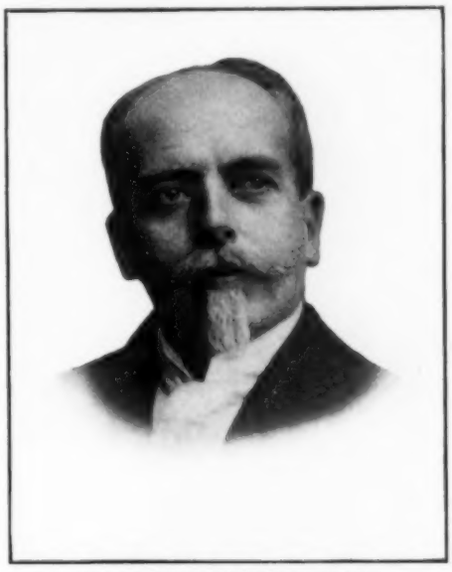
Throughout the mill room safety switches (Fig. 2) are installed and these are arranged in series electrically so that the clutch will be disengaged and the brake simultaneously applied, if a safety bar, one of which is provided for each mill, is tripped. In the motor room there is provided a single push button for engaging the clutch, and the clutch is energized by means of an automatic rheostat in such a way as to make it pick up the load gradually and at the same rate of speed for each start.

One novel feature of the electric control system is that if the clutch is disengaged by the operation of any one of the safety stations located throughout the mill, as stated above, the brake is simultaneously applied. If, however, the clutch is disengaged by means of a stopping button on the motor-control panel, the brake is not applied but the load is allowed to coast to rest. Consequently, the quick stop is obtained only in case of an accident.

Another feature of the control system is that when the safety stations have once been tripped, they can be reset immediately, but power cannot be put back on the mill line until the single starting button located on the motor-control panel has been operated. A slack cable switch is also provided so that in case of breakage of the rope drive the clutch is automatically disengaged and the brake applied.

Obituary

James Mapes Dodge, chairman of the Board of the Link-Belt Company and a pioneer in the development of conveying and elevating machinery, died at his home in Philadelphia on Dec. 4, 1915. He was born June 30, 1852, at Waverly, N. J., and received his education at Rutgers College and Cornell University. After a varied experience in the East, partly with shipbuilders, he went to Chicago, where he joined with William D. Ewart,



THE LATE JAMES M. DODGE

inventor of the Ewart link-belting, in the development of the chain business. At that time, about thirty-five years ago, the application of chains to power transmission was exceedingly limited, and their use in elevating and conveying machinery was practically unknown. Later Mr. Dodge came to Philadelphia and formed a partnership with Mr. E. H. Burr, to handle the Ewart chain. Out of this partnership the Link-Belt Engineering Company grew in 1888. In 1889 he developed the conical pile system of storing coal, for which he received the Elliott Cresson gold medal in 1907 from the Franklin Institute. He designed the links and attachments for all of the link-belts now used by the Link-Belt Company.

Fertility of invention and mechanical ingenuity combined with a high order of executive ability were the qualities which brought Mr. Dodge his success. He was for many years closely associated with the late Frederick W. Taylor, in the accomplishment of remarkable results for the benefit of both employers and employees, in a wide circle of industries. His influence was always towards the development of self-help, initiative, ambition, and responsibility in the men, and of a spirit of fair play and humanitarianism as real business assets. Mr. Dodge was a past president of the American Society of Mechanical Engineers, a vice-president of the Franklin Institute, and a member of various other clubs and societies.

Personal

Mr. **H. W. Hardinge** has been awarded the John Scott medal by the Franklin Institute of Philadelphia for the invention of his conical mill.

Mr. **S. A. Ionides** of Denver is on a professional trip which will take him through California and the Northwest.

Mr. **R. J. Johnston** has been appointed superintendent of the Donora zinc smelter of the American Steel & Wire Company.

Mr. **W. J. Lakeland** of the Burma Mines, Ltd., has arrived in this country from Burma for a visit to the mining and metallurgical centers of the West.

Mr. **F. W. McNair**, president of the Michigan College of Mines, has been on a trip through the Western States, traveling to the Pacific Coast.

Mr. **Benjamin Magnus**, formerly general manager for the Mount Morgan Gold Mining Company, Ltd., of Mount Morgan, Queensland, has opened an office as consulting engineer at 61 Broadway, New York City.

Mr. **Ernest B. Pearce** will have charge of the new tin smelting plant of the American Smelting & Refining Company at Perth Amboy, N. J. Mr. Pearce has had considerable experience in tin smelting in other countries.

Mr. **F. Peiter** has severed his connection as coke oven engineer with the Gas Machinery Company, Cleveland, Ohio. Mr. Peiter was formerly connected with H. Koppers Company, as designing and constructing engineer, and also with the Tennessee Coal & Iron Company.

Mr. **W. R. Rust** has resigned the presidency of the company operating the Tacoma smelting plant, which is controlled by the A. S. & R. Co., and will retire from the management of the property.

Mr. **W. L. Saunders**, president of the Ingersoll-Rand Company and member of the Naval Consulting Board, gave a talk on "Preparedness" at the annual banquet of the Plainfield, N. J., Board of Trade on Dec. 2.

Mr. **E. A. Cappelen Smith** has been elected a director of the Chile Copper and Chile Exploration companies. Mr. Smith is at present in Chile.

Mr. **W. E. Tracy**, who for some years past has been superintendent of the mill of the Liberty Bell Company at Telluride, Col., has severed his connection with the company and has gone to Plainfield, N. J.

Book Reviews

Electrothermic Smelting of Iron Ores in Sweden. By **A. Stansfield.** Bulletin 344, Mines Branch, Department of Mines, Canada. Ottawa: Government Printing Bureau.

This monograph of 67 pages is a most timely publication. It narrates the observations of Professor Stansfield of McGill University (well-known for his valuable book on *The Electric Furnace*), while on a vacation trip in Norway and Sweden in 1914. The data given were accumulated and are discussed with particular reference to Canadian conditions and possibilities.

Interesting points brought out are: The Swedish electric furnaces make charcoal pig iron cheaper than Swedish charcoal blast-furnaces; the thermal efficiency of utilization of the electric energy is 68 to 74 per cent; the system of gas circulation is cumbersome but does not increase the output per kw. year; the cost of making charcoal pig iron in Canada, with power at $10 per hp. year would be $20 to $21 per ton, of coke iron, $18.50 to $19.50. At present, the author sees prospects in Canada only for the electric production of high-grade charcoal pig iron.

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The Sampling and Chemical Analysis of Iron and Steel. By **O. Bauer** and **E. Deiss.** Translated by **W. T. Hall** and **R. S. Williams.** Octavo, 373 pages, 135 illustrations. Price, $3.00 net. New York: McGraw-Hill Book Co., Inc.

The translators have rendered a distinct service to their chemical colleagues in translating this fine work. Professor Bauer wrote the 125 pages on sampling, and Mr. Williams translated it; it should be read by every man entrusted with taking samples of iron or steel for analysis. Mr. Deiss wrote the rest of the book, on methods of chemical analysis, including in it only the methods found reliable at the German Royal Testing Bureau, Gross-Lichterfelde, Berlin, and omitting rapid "works methods." The translators have amplified this part by adding about 25 quick methods, mostly of American origin, including also, Walker and Patrick's test for total oxygen. It is a keenly practical book, for the use of expert analysts anxious to get results of high reliability and accuracy.

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The Rare Earth Industry. By **Sydney J. Johnstone.** Octavo, xii + 136 pages; 42 illustrations. Price $3.00 (7s. 6d. net). New York: D. Appleton Co.

The principal chapters of the book are on thorium and cerium, titanium, zirconium, tantalum and niobium, tungsten, uranium and vanadium. The contents also include the manufacture of incandescent mantles, pyrophoric alloys and electric incandescent lamps; a final chapter of 18 pages, on "The Industry of Radioactive Substances," was written by A. S. Russell.

The treatment is principally technical and commercial with special emphasis on statistics of production and cost. Each chapter gives, regarding the rare element, a fairly complete bibliography, its occurrence, production, treatment, uses and methods of analysis. The field covered is so extensive and varied that the author could not have first-hand information on every topic, as a necessary consequence some of the statements (*e.g.* on pyrophoric alloys) are not as reliable as are those drawn directly from Mr. Johnstone's experience. But in the line of the mantle industry and the manufacture of electric lamp filaments the descriptions of commercial practice are particularly complete and satisfactory. Mr. Russell's chapter on Radioactive Substances is also cleverly written. The book, as a whole, contains an unusual amount of scarce information.

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Chemical Constitution and Physiological Action. By Prof. Dr. **Leopold Spiegel.** Translated from the German by **C. Luedeking** and **A. C. Bolston.** 12mo, 156 pages. Price, $1.25. New York: D. Van Nostrand Co.

This is a serious attempt to collate what is known of the relations between chemical constitution and physiological action, in order to provide a basis for rational scientific medical treatment. After a chapter of general considerations, in which the subject is discussed from a very broad point of view, there are 20 pages given to the consideration of various inorganic compounds and the rest of the book to organic—20 pages to the aliphatic series, 17 pages to the aromatic series, and 70 pages to nitrogen compounds. The odor and taste of these medicines are not considered, but only their medical effects as related to their chemical structure. The work was well worth translating for the benefit of medical students and progressive practitioners.